Labahn / Kohlhaas

Cement Engineers' Handbook

BAUVERLAG

Cement Engineers' Handbook

Originated by Otto Labahn

Fourth English edition

by B. Kohlhaas

and

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Publisher's foreword

Since the publication of the first edition of "Cement Engineer's Handbook" 28 years ago, this book has gained an established reputation as "Labahn" in the cement industry. In its conception it has survived its original author. In form and contents it has become an entirely new book, however. This change reflects the great technical developments that have taken place in cement manufacture in the intervening years.

The first edition was, with the exception of the chapter on quarrying, written entirely by Otto Labahn. The fully revised fourth German edition of 1970 was still within the range of one individual author, Wilhelm Andreas Kaminsky, who undertook the revision. When it was decided to produce the present sixth edition, it soon emerged from the preliminary discussions that in this age of specialization the preparation of the new text for a book of this scope would have to be entrusted to a team comprising authors from a wide variety of technological disciplines associated with cement manufacture.

In this effort we have been fortunate in having had the services of Bernhard Kohlhaas as editor, co-ordinator and author.

He proved indefatigable in seeking suitable co-authors for this project and he himself undertook the revision of a number of the manuscripts supplied. These duties made greater claims upon his time and attention than had been expected. We are indeed grateful to him for his unflagging devotion to the task.

The guiding principle of this new edition is the same as that which Kaminsky enunciated in the preface to the edition which he had revised:

The subject matter of the book as a whole corresponds approximately to the range of problems which concern the engineer engaged in present-day cement manufacturing practice. The guiding principle remains: to present all that is essential and important in a conveniently assimilable form. At the same time, this approach rules out any very detailed treatment of individual subjects.

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Contents

A. Introduction

By B. Kohlhaas

The first edition of the Cement Engineers' Handbook was published in 1954. Up to that time no such reference book for the engineer or technician in cement works practice had been available. Although four subsequent editions appeared, the demand for the book continued as brisk as ever. The major developments that had meanwhile taken place in the cement industry in Germany and other countries justified the decision to produce an entirely new edition that would take due account of the latest cement manufacturing technology.
The text for this new edition has been written by a team of experts in their respective fields of specialization relating to cement manufacture and the machinery used at all stages of the process. Some of the chapters have been substantially enlarged and updated from those contained in the earlier editions of the Handbook. A number of new chapters have moreover been added. The entire subject matter has been extensively recast and rearranged, as will be apparent from the comprehensive table of contents. Each chapter is accompanied by a list of literature references enabling the reader to consult more detailed published information on matters of particular interest to him. The names of the authors are given at the beginning of the chapters.
The following information on the sections and chapters into which the book is divided will help the reader to understand its layout and to use it with greater convenience.

B. Raw materials

I. Geology, raw material deposits

This section is of especial significance in connection with setting up a new cement works and ensuring a long-term supply of good-quality raw materials.

II. Quarrying the raw materials

The modern techniques of winning the raw materials by quarrying or mining operations are described. The restoration of worked-out quarry sites in the interests of landscape conservation also receives attention.

III. Raw materials storage

The raw materials needed for cement manufacture are seldom found in the ideal chemical composition in their natural state. Besides, quarrying operations usually stop at the week-ends, whereas cement production proceeds continuously. To cope with the high production rates of modern cement plants and keep them supplied with materials, capacious intermediate storage facilities are required, so as to make the plants independent of the quarry operating rhythm.

C. Cement chemistry – cement quality

After presenting a historical introduction, the author of this section deals in detail with the cement raw materials, their suitability and the calculation of the raw mix proportions. The chemical, mineralogical and physical processes associated with burning the materials in the kiln are described.
Portland cement clinker and the assessment of its quality are discussed. Other sections deal with cement grinding, storage and hydration. The types and strength classes of cement, as well as cement testing procedures and associated matters, are also considered. Finally, some information on standard specifications for cement in various countries is given.
These matters are dealt with much more fully than in earlier editions of the Handbook, with the object of giving the mechanical and electrical engineers (including those concerned with process control and instrumentation) in cement manufacture a better understanding of the problems involved.

D. Cement manufacture

This chapter is devoted to the actual process of making cement. The various stages are described. The wet process and the shaft kiln are only briefly considered. On the other hand, the dry process with raw meal preheating and the precalcination principle are treated in some detail, as are the preparation of the raw materials, the storage and homogenization of the raw meal, and the cooling of the cement clinker.
This latest edition of the Handbook moreover contains up-to-date information on firing technology, kiln systems and refractory lining construction.
Clinker storage now has a separate section allotted to it. In view of today's concern with environmental pollution prevention, the dust-free storage of large quantities of clinker is very important.
Present-day methods of packing and despatch loading are described (Chapter E).
Whereas the subject of materials handling and conveying (Chapter F) was rather summarily dealt with in earlier editions, it has now received much more detailed treatment. Feeding and proportioning are also included.
Process engineering and automation are of such importance in modern cement manufacturing technology that they have a separate chapter devoted to them, in which the principal aspects are considered in some detail (Chapter G).
The subjects of environmental protection and industrial safety (Chapter H) are now likewise fully dealt with in the Handbook for the first time. These are subjects of great importance in connection with modern cement manufacture, which indeed can be carried out only if the statutory and other requirements relating to them are duly complied with.
The book contains some further chapters devoted to various matters that concern the cement works engineer: maintenance and wear; workshops and spare parts store; water supply, compressed air; personnel requirements; lubricants; firefighting equipment; laboratory equipment.

B. Raw materials

I. Geology, raw material deposits, requirements applicable to the deposit, exploration of the deposit, boreholes, evaluation of borehole results, calculation of reserves

By H.-U. Schäfer

1 Raw materials and quarrying methods

The raw materials for cement manufacture which are the subject of geological exploration are mainly limestones and clays. In the geological sense both are sedimentary rocks which may occur as hard or dense material (commonly known as "rock") or softer soil deposits. They may be of any geological age. Limestones mostly occur in the form of rock, sometimes constituting whole mountainous formations. In Europe, more particularly the Devonian granular limestones, the Jurassic and Triassic limestones of the Alpine region and the Cretaceous limestone deposits are of importance.

Whereas the limestone deposits of the Precretaceous period are usually composed of fossil limestones which in many instances were subjected to metamorphic change (e.g., marbles, siliceous limestones), the younger and mostly Postcretaceous limestones occur both as fossil deposits and as limestone-clay mixtures. The latter are referred to as lime marl (calcareous marl) or marl, depending on the limestone/clay ratio of the mixture (see Duda, Vol. 1, Section 1). These limestones also include the so-called natural cements in which CaO, SiO_2, Al_2O_3 and Fe_2O_3 are present in such proportions that the lime standard is around 100 and the desired moduli can be obtained by the addition of only small quantities of corrective materials. Such deposits are, however, or rare occurrence.

The youngest recent and sub-recent limestones include coral limestones, which occupy in some cases an intermediate position between (consolidated) rock and unconsolidated material. Deposits of shells, which can also be used in the manufacture of cement clinker, belong to the last-mentioned category.

The clay mineral component used for cement manufacture will generally be a soft or loose-textured material: clays, silts, or sands with high content of clay minerals. These materials are classified according to particle size distribution rather than mineralogical composition (Table 1). Rock-type clay materials may occur as clay slate, shale and (to some extent) crystalline slates. Subject to chemical suitability, such rocks as granites, gneisses, basalts and basaltic tufas or pozzolanas may also serve as clay mineral components.

Additive materials for clinker production may be needed for correcting the chemical composition of the raw mix, e.g., materials providing Fe, SiO_2 or Al_2O_3, more particularly the most inexpensive ones that can serve the purpose, e.g., roasted pyrites or low-grade iron ore, laterite, quartz sand or quartziferous weathering products of metamorphic rocks, and bauxite.

Table 1: Nomenclature of clay, silt, etc. in accordance with particle size distribution (DIN 18123)

clay	< 0.002 mm
silt	0.002 – 0.063 mm
sand	0.063 – 2.0 mm
gravel	2.0 – 63 mm
stones	> 63 mm

Table 2: Limits imposed on the MgO content of portland cement materials by Standards in various countries (according to Cembureau, 1968)

Country	max. % MgO by weight
Rumania	2.5
Belgium, Denmark	3
Italy, Mexico, New Zealand, Pakistan, Portugal, Great Britain	4
Australia	4.2
Bulgaria	4.5
Argentina, Austria, Canada, Chile, Cuba, Finland, France, German Democratic Rep., Fed. Rep. of Germany, Greece, Hungary, Indonesia, Ireland, Israel, Japan, Netherlands, Norway, Poland, South Africa, Spain, Sweden, Switzerland, Taiwan, Turkey, USSR, Venezuela, Jugoslavia, People's Rep. of China	5
Brazil, Czechoslovakia, India, USA	6

The assessment of the suitability of the raw materials for cement manufacture is based chiefly on their chemical composition. For limestone components the so-called lime standard is used as a criterion, giving information on the CaO content as well as on the "hydraulic" constituents SiO_2, Al_2O_3 and Fe_2O_3. It is in any case preferable to assessing the materials merely on the basis of CaO content.

The rocks to be used as clay mineral components can most suitably be assessed by calculation of the silica ratio and the alumina ratio.

For deciding on the suitability of raw materials it is furthermore essential to perform mix proportioning calculations in order to ascertain the content of alkalies, sulphates, chlorides and MgO introduced into the raw mix.

The permissible limit values for the content of sulphates, alkalies and chlorides must be conformed to.

The content of magnesium that can be permitted is laid down in standards which vary from one country to another (Table 2). It will have to be decided in each particular case whether anything in excess of the standard specified content can be allowed, since there are no suitable raw materials that fulfil the requirement of, in most cases, not exceeding about 4 – 5% MgO (by weight) in the cement. Under certain circumstances, too, infrastructural or economic reasons may constitute a deciding factor in justifying a departure from the standard limit.

Exploration of limestone and clay deposits for cement clinker manufacture has three aims:

(1) verifying the quality of the raw materials;

(2) establishing the range of variation in quality of the raw materials throughout the working life of the deposit;
(3) verifying the workable reserves of raw materials.

For the technological planning of the machinery for a cement manufacturing plant it is of major importance to ascertain the ranges of variation of individual raw material constituents in the deposit throughout the operating life of the plant, for only in this way can trouble-free operation yielding a final product of good quality be ensured. Variations of relatively short duration, ranging from months up to about half a year, should also be known in good time, so that suitable precautions in terms of machinery and process technology can be taken or otherwise, in the light of economic considerations, corrective ingredients that will help maintain a product of unvarying quality can be quarried or purchased.

Exploration for limestone and clay mineral components for cement manufacture mainly comprises geochemical investigations, though the bedding conditions of the deposit also play an important part with regard to subsequent planning of the quarrying operations to meet the raw material requirements of the cement works.

Besides qualitative conditions, the deposit will also have to fulfil quantitative conditions more particularly in connection with the method of quarrying or digging to be employed.

Cement works with clinker outputs of between 1000 and 6000 t/day need a raw material input of 2000 to 12000 t/day (assuming clinker production on 330 days and quarrying operations on 260–280 days per year), about 50–90% of this quantity being limestone and 10–50% clay mineral material.

2 Exploration

2.1 Exploration procedure

The exploration procedure will always have to be suited to the particular conditions of the deposit under investigation, so that it is here not possible to give more than a general outline description.

Generally speaking, the exploration of cement-grade deposits will comprise three stages:

Stage 1: Field inspection of a number of deposits, surface tests, a limited number of exploratory borings (including core borings, if necessary), simple hydrological and tectonic investigations, large-area mapping.

The object of this first stage of an exploration, which can be referred to as reconnaissance prospecting, is to select one or more deposits for further detailed prospecting. In this connection the quality of the deposit is especially important, while problems of mining or quarrying are given comparatively little attention at this stage.

Stage 2: On completion of the first stage, one or more deposits are selected for detailed investigation. On the basis of a comprehensive drilling program the

deposits are broadly studied with a view to ascertaining their chemical characteristics over extensive areas. In conjunction with the borings, further investigations are carried out for determining the bedding conditions, ground water and possibilities of working the deposit, the object being to assess the suitability of a site for quarrying or open-cast working. More particularly, the second stage aims to find the most suitable area for siting the quarry or to select the most favourable of two or more deposits potentially available for supplying the raw materials.

Stage 3: This is the stage of detailed exploration, using a grid of closely spaced boreholes for the purpose of determining chemical properties of the raw material components and their variations over short distances, in order to gear the process engineering design of the cement works to these conditions.

Furthermore, special investigations for planning the quarrying operations are carried out. The structure of the deposit is studied in detail. In addition, the possibility of working the material by ripping may, for example, be examined. While these exploratory operations are in progress, assessment of the results already available is undertaken, so that any problems emerging therefrom can be fed back to the exploration work and duly taken into consideration. On completion of the third stage of exploration, the deposits are fully known as regards their qualitative, quantitative and mining or quarrying engineering features and can be got ready for opening-up.

2.1.1 Trial pits and surface samples

Taking samples from a trial pit is usually a form of surface testing, because it is not possible economically to dig shafts of any great depth into limestone rock.

On the other hand, with clay soils it is possible to base the exploration on a comprehensive grid of test shafts. However, if the clay deposit is of substantial thickness, it is better to use drilling techniques, as the digging of deep shafts is very expensive.

Mostly a combination of the two methods is adopted.

With limestone, pits are dug in places where the solid rock is covered by other material which has to be removed in order to expose the limestone for testing. Such exploration also affords an opportunity of testing the overlying material and assessing its possible usefulness.

When the surface of the rock has been exposed by excavation, or if it occurs as an outcrop, material for examination can be sampled in two ways: either as spot samples from a locally limited area of exploration or as continuous samples taken along a line (or a long exploration trench) extending at right angles to the strike.

With continuous sampling it is important that the samples should be properly representative of the rock strata under investigation. This can most simply be achieved by excavating a cut from which, for approximately unvarying cross-section, a constant quantity of sample material per unit length is obtained.

If a cut is too expensive or indeed impracticable, it will alternatively be necessary to take from the strata in question a sample quantity which bears an appropriate relation to their depth and extent.

When a trial excavation is made, sampling and testing should, as far as possible, not be confined just to the surface of the limestone, but should extend down to at least below the top weathered layer of rock. In most cases this will require the aid of a heavy excavator or rock breaking hammers and a compressor. In young chalk limestones or coral limestones a ripper or even lighter equipment may suffice for the purpose.
In any case it must be investigated whether the limestone is liable to undergo changes in its chemical character as a result of atmospheric influences, weathering, circulating underground water, or ground water occurring close to the surface. In the last-mentioned case the chemical properties of the ground water are also of considerable importance.
If clay occurs in the form of a loose-textured soil-type deposit, exploratory excavations (trial pits, etc.) can be made with simple means. The stability of the walls of such excavations should be given due attention in view of the danger to men working in the excavation, or to machines standing at the edge thereof, arising from a sudden collapse of a wall. If necessary, timbering will have to be installed.
The arrangement of trial pits and trenches in clay is similar in principle to that in limestone. The same is true of the sampling procedures.
It is advantageous to have hermetically closable jars or canisters available for storage of the rock or soil samples with their in situ moisture content, because more particularly with clays the moisture conditions are important in deciding what type of preparatory processing machines will have to be used. Where excavating machinery is used for digging the trial pits, the experience thus gained can provide useful indications with regard to the planning of the future quarrying operations (lumpiness, stickness, distintegration, suitability for excavation by means of power shovels, wheel loaders, etc.).

2.1.2 Drilling

The selection of the most suitable drilling or boring method in terms of technical suitability and also of economy is the fundamental condition for successful exploration. In the main, there are three drilling techniques to choose from: solid-bit drilling with removal of the cuttings by circulating water or other flushing medium; core drilling with continuous core extraction; percussive rotary drilling with removal of cuttings by means of compressed air.
Solid-bit drilling with rotary bits and removal of cuttings with the flushing medium is suitable only in exceptional cases for exploratory drilling in solid rock deposits. If this method is used, it should be known in advance whether it will not cause changes in the chemical character of the samples, e. g., by the dissolving of soluble compounds (alkali chlorides, for example) or by failing to reveal the presence of marl strata or clay enclosed within the rock under investigation.
Similar considerations are applicable to percussive rotary drilling with crawler-mounted machines of the type used for the drilling of blastholes. This method is unsuitable for deposits consisting of loose-textured or soil-type deposits.
Core drilling is the most reliable method of obtaining samples for assessment. In this technique a continuous core is extracted over the full depth of the hole, so that,

if the drilling operations are carried out by suitably experienced personnel, the geologist can obtain full information of all details of the limestone deposit at all levels below the surface.

2.1.2.1 Core drilling in limestone

For successful exploration with the aid of core drilling the correct choice of drill bits, core barrels and flushing media is of major importance.
For core borings in limestone the diameter should be not less than 75 mm. With smaller core diameters there is a risk that jammed cores will pulverize thin soft intermediate strata, that the hole will be choked by caving and that material from some strata may be removed along with the flushing medium.
An upper limit to the core diameter is imposed by considerations of economy. Diameters of 120 mm and upwards are seldom used, except under critical conditions where drilling has to be done with water flush in porous rock and, by employing a large diameter, washing-out of soluble compounds can be prevented at least in the interior of the core. On the other hand, cores which are too small will make the evaluating geologist's task more awkward, while the halves into which the core specimens are split for the purpose of possible supplementary or follow-up tests are then rather unsuitable for the purpose.
The choice of a suitable drill bit will depend on the rock itself: the bedding, fissuring and tectonic characteristics of the deposit, and the abrasiveness of the rock. Carbide-tipped as well as diamond drill bits are used. With large diameters and heavily fissured rock the risk that parts of the core will tilt and jam in the core barrel is greater with carbide bits; besides, the core is more exposed to the action of the flushing medium than with diamond bits. In such cases the choice of the most suitable bit will depend on the foreman-driller's experience.

2.1.2.2 Core barrels

Three types of core barrel are available from which to make a choice:
the single tube, the double tube and the wire line type. In addition, there are special types of barrel, which may have to be used under exceptionally difficult conditions.
The three types are illustrated schematically in Fig. 1. The single tube barrel is provided, near its bottom end just above the bit, with a core catcher ring which grips the drilled core during extraction of the drill rod and thus prevents it from dropping down the hole. The basic condition for successfully using the single tube barrel is that the rock is of such a kind (massive and uniformly strong) that a core can indeed be drilled from it. If the limestone is composed of thin plate-like strata or if it easily disintegrates during drilling, there will be a risk that part of the core will fall back into the hole on extraction. Furthermore, in such cases the geological and geochemical assessment and analysis of the sample is rather difficult, since the sample consists merely of fragments which make it impossible to carry out all the necessary observations in detail. Another and very serious drawback of the single tube is that the core is enveloped in a flow of flushing medium along its entire

length, so that, especially if water flush is employed, fine stone chippings and any sandy, silty or clayey inclusions are likely to be washed out.
With the double tube type of core barrel the inner tube is connected through ball bearings to the outer tube and therefore does not revolve with the latter (which carries the drill bit). In this way the core remains at rest and thus substantially undisturbed. The most important advantage of the double tube, however, is that the core is not enveloped in the flushing medium, which is, instead, forced through the annular space between the inner and the outer tube. The core comes into contact with the flushing medium only at the lower end of the barrel, where the inner tube terminates and a gap for the passage of the medium exists between the two tubes. Because of this limited area of contact, very little of the core is washed out, though of course some dissolving of soluble constituents in this area cannot be avoided.

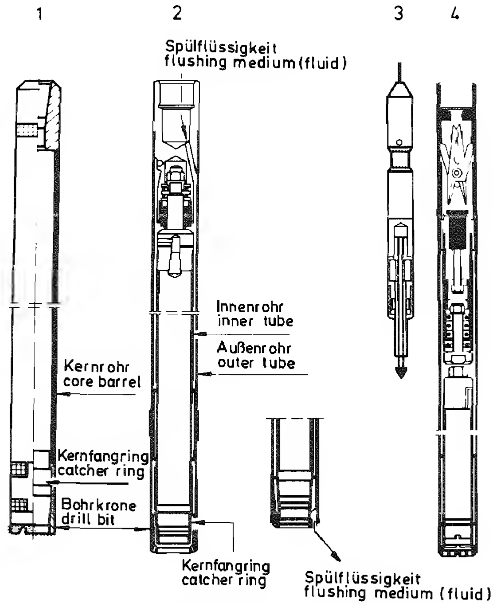

Fig. 1 : Types of core barrel : single tube barrel (1), double tube barrel (2), grapple device (3) with wire line barrel (4) (based on information from Atlas Copco)

Special double tube core barrels are equipped with bits which are so designed that the flushing medium does not emerge from the gap between the inner and the outer tube, but is discharged to the outside before or within the cutting edge of the bit. Inside the bit (Fig. 1) the inner tube is in such close contact with it, that practically no water can get to the core sample.
If borings are carried out in very soft and shattered material (though firm enough to enable a stable hole to be drilled), it is possible to use a special double tube core barrel in which a third tube, made of plastic, can be inserted into the inner tube. The core is then removed together with the plastic tube from the barrel, so that a substantially undisturbed sample for assessment is obtained.
If the deposit consists of material in which it is not possible drill a stable hole even with mud flush, a wire line barrel can be used.
With the wire line barrel the whole drill rod is of the same diameter as the core barrel itself. The inner tube, however, is not permanently connected to the outer tube by ball bearings, but is gripped in it by means of a catch mechanism. When the length of core corresponding to the length of the barrel has been drilled, a wire line with a kind of grapple is lowered into the hole and releases the catch, enabling the tube containing the core sample to be drawn up. This procedure offers the advantage that the drill rod need not be extracted in order to extract the sample from the hole, so that the risk of caving and blockage of the hole is obviated. Besides, the operation of extracting the core tube takes less time than it does with the other systems. There are also special wire line core barrels in which the flushing medium emerges before the cutting edge of the bit, so that there is hardly any contact between the core and the medium.

2.1.2.3 Flushing media

The choice of the flushing medium for borings in limestone is of major importance in connection with the subsequent geochemical investigation of the samples.
It has already been noted that with a fluid medium for flushing the borehole there is a risk that clay and marl strata, as well as sand and silt inclusions, will be washed out and that soluble constituents of the limestone will likewise be lost. In principle, a distinction is to be drawn between air and liquid flushing media. In all cases air flush is preferable, because it ensures that no constituents will be removed by washing or dissolving action. With air flush it is often unnecessary to use a double tube core barrel, for in the single tube the samle is enveloped only in a stream of air, though admittedly the rate of drill bit wear is then higher.
With water flush the pressure of the water should be kept as low as possible. The higher the pressure, the greater is the risk of disturbing the sample by washing out some of the material. For the purpose under consideration water is the only suitable liquid flushing medium or otherwise only such media whose constituents can afterwards, in the chemical analysis of the rock samples, unambiguously be identified as having originated from the flushing medium.
In connection with water flush, the porosity of the limestone is of major importance. In any case the water used for the purpose should be analysed to make

it possible subsequently to draw conclusions as to any effect that it may have had on the samples. For example, if salt water is employed, it will in any case be difficult to distinguish between the alkali content of the limestone and the alkali introduced with the flushing water.
In highly porous limestone which can be suspected of having a high content of alkali, chlorine and sulphate the core drilling technique with air flush is the only possibility of obtaining suitable samples for geochemical investigation.

2.1.2.4 Core drilling in clay

If the clay mineral component for cement manufacture occurs in the form of a solid rock (shale, slate, etc.), the same drilling techniques as for limestone can be applied. However, if it occurs as non-cohesive soil, other methods will have to be chosen. In such cases, as a rule, percussive drilling will be used and the hole will be cased as drilling proceeds, so as to prevent caving on extraction of the rod. The sampling device used in borings of this type is usually a spoon sampler which, on being extracted, closes its lower end and thus prevents the soil sample from falling out. The sample obtained in this way is disturbed, however, so that the information it gives on bedding conditions, fissuring, etc. may be questionable.
This technique can also be applied to cohesive soils, but in such soils it is alternatively possible to use a rotary drill, equipped with a carbide-tipped bit. If undisturbed samples are required, a core barrel of the double tube type can be used. In many instances, however, a single tube core barrel will adequately serve the purpose if water flush can be dispensed with. Drilling operations are liable to be particularly difficult, even if little water is used, in clays containing minerals which swell and thus cause a narrowing of the hole. Under such conditions it is certainly necessary to case the hole directly above the drill bit.
Drilling in loose-textured or friable material should, if at all possible, be performed without a flushing medium.
In especially difficult cases the drilling operations may be carried out with double tube core barrels or wire line barrels equipped with a plastic inner tube for enclosing the sample. The plastic tube is withdrawn along with the sampled material and serves also as its container for despatch to the laboratory.

2.1.2.5 Treatment of the cores

The cores extracted from the boreholes are stored in boxes. If they are to be transported as freight over long distances, the boxes should be made of suitably strong material and strengthened with metal. Cores obtained from loose-textured deposits should additionally be protected in plastic bags.
In the field, the cores should be recorded by the geologist directly after their removal from the core barrel. Such records can most suitably be supplemented by colour photographs of each core. Fields records should be as comprehensive as possible so as to enable the samples also to be correlated with any supplementary borings that may be made later or with the actual conditions encountered on

opening-up the quarry. The drilling report should contain technical data relating to the drilling operations and also geological data, so that, when the geochemical tests results become available, a complete diagram for each borehole is obtained. Each report should contain information on the location, altitude of the starting point and designation of the borehole. For each drilling depth, the diameter of the hole, the type of core barrel, the type of bit and change of bit, amount of core recovered, flushing losses and rate of drilling progress should be noted. With the aid of this information it will, in the event of subsequent additional investigations, be possible to discuss whether drilling can be done more easily and cheaply with different equipment. Furthermore, the foreman-driller should keep a record of the ease or difficulty with which the rock can be drilled. Although this is a matter of subjective judgment, it can facilitate the work of correlating the profiles in rock of a macroscopically very uniform character.
The correct geological description of the samples comprises the designation of the type of rock penetrated, the colour of the rock, its granularity, information on inclusions of foreign rock or mineral inclusions, porosity and hardness, bedding, fissuring, and information on any faults encountered.
Furthermore, each drilling report should record the samples taken from the core drilling run, unless the core is divided and one half is retained for possible future reference. If information on approximate stratigraphic classification is available, this too should be included in the report. Under certain circumstances, field tests may be performed on the cores in order to check the $CaCO_3$ content or the suspected presence of MgO. The results of these tests are likewise to be added to the report. A graphic representation of the conditions encountered is in any case necessary.

2.1.2.6 Testing of drilled cores

For the purpose of testing, the cores are divided into sections on the basis of macroscopic criteria. Each section is then subdivided into portions for analysis, with due regard to the method of quarrying to be employed. In the case of a relatively thin deposit, i.e., of limited depth, which will have to be worked by ripping (or if ripping has to be applied for other reasons), the length of the analysis portions should not exceed twice the ripping depth.
On the other hand, if benching is to be employed, the portions for complete analysis should not be more than 5 m long.
If at all possible, the core should be divided in halves, one half being retained for future reference, while the other is sent to the laboratory. Cores of very large diameter may also be quartered.
If such division of the core is not possible, the whole core must be despatched to the laboratory, where it may have to be comminuted by crushing. In such cases the core portions should not exceed 1 m in length, in order to keep down the cost of analysis (see below).

2.1.2.7 Rotary percussive drilling with crawler-mounted machines

To supplement the core borings and to fill in the network of boreholes in solid rock deposits, additional drilling can be carried out inexpensively with the aid of a crawler-mounted drilling machine, of the type used also for the drilling of large-diameter holes for blasting.
The drill bit, operating by rotary percussive action, shatters the rock, and the cuttings are removed from the hole by air issuing from the bit.
The dust carried out of the hole with this flushing air can be trapped in a dust collector, which is mounted on the drilling machine. It comprises a cyclone in which the coarser particles are precipitated, while the finer ones are retained in special filters. The suction extractor is connected to a flexible tube which terminates in a plastic sleeve forming an airtight closure over the mouth of the borehole, so that all the dust can be collected. For testing the samples it is important not only to analyse the dust precipitated in the cyclone, but also to include the fine particles trapped in the filter equipment.
With borings of this type it often occurs that the dust is collected without the aid of a suction extractor, merely by placing a sheet of plastic around the top of the hole and collecting the dust, discharged from the hole, on this sheet. This method is to be condemned, unless the object of such borings is merely to obtain approximate guiding data or if it is desired, quickly to obtain details of the chemical composition at one particular point in a deposit on which reliable information is already available.
Clay intercalations, sand inclusions or soft moist limestone strata are forced aside by the rotary percussive drill bit and remain sticking to the wall of the borehole, so that a proper sample of such material is not obtained. Nor is it possible to get information on the presence of any cavities in the rock. The most serious drawback of rotary percussive drilling, however, is that it offers no possibility of sampling the rock as such and thus forming a reliable picture of the occurrence of limestone in the deposit under investigation.

2.1.3 Stratigraphic investigations

In prospecting for raw materials for the manufacture of cement only secondary importance attaches to stratigraphic investigations, because the suitability of the raw materials depends mainly on chemical features and is not confined to any particular geological age.
Accordingly, stratigraphic investigations are usually limited to macroscopic classification of the drilled cores and to assigning characteristic datum horizons for correlating the individual core borings.
More important, on the other hand, is the chemostratigraphic examination of the borehole profiles, especially if the deposit appears to be of a very unvarying character on the evidence of field observations and of the cores.
Quite often it is only in this way that differences in facies are ascertainable which would otherwise remain undetected. Such differences may nevertheless be of considerable importance in connection with the subsequent planning of the

quarrying operations, e.g., if the average CaO content of the limestone is only about 46% and there is a marked shift to lime marl facies.

2.1.4 Tectonics

Of greater importance than stratigraphic investigations in the present context are investigations on the bedding conditions and structure of the deposit. The precise interpretation of these factors constitutes the basis for the reliable geochemical evaluation of the results of the borings and for planning the quarrying procedure.

2.1.4.1 Limestone deposits

The investigation begins with surveying the available exploration points relating to the deposit. The bedding features and any faults affecting them can be observed and measured there. Particular attention should be paid to "micro-tectonics", i.e., the structural characteristics and their variations within distances of the order of a few metres or indeed of decimetres, since such characteristics can be of major importance in determining the alignment of the quarry face. Furthermore, the exploration points provide information on the presence of any strain zones which manifest themselves in variations in bed depth or which have caused foliation of the limestone.
Fracturing and faults which extend as more or less straight planes through the limestone are important in connection with further planning. Young limestone deposits, in particular, are often penetrated by such fractures whose faces are often crusted with calcite and coated with a thin layer of clay. Such planes should receive particular attention in quarry planning, because ground vibrations due to blasting are liable to cause subsequent rock slips along these planes, resulting in sudden collapse of large portions of the quarry face.
If the exploration points available for the deposit are not sufficient to permit complete mapping of its structural features, photogeological mapping may be helpful, provided that aerial photographs in the scale range from 1:5000 to 1:15000 are obtainable and the vegetation on the terrain does indeed allow photogeological interpretation.
Another valuable aid in assessing the structural conditions of the deposit is provided by the results of borings. For these, correlation can be based primarily on the stratigraphic description of the individual borings. Such correlation must not wait till the drilling operations have been completed, but should proceed at the same time as those operations, in order to monitor and, if necessary, correct the locations chosen for the further exploratory boreholes in the light of the structural assessments.
Interpretation of the macroscopic stratigraphic core drilling records is linked to profile sections along the network of boreholes and to maps indicating the depths at which particular stratigraphic horizons occur. In this way a good idea of the structure of a deposit can be obtained, which can be supplemented with the results of geochemical investigations.

The chemical data of each borehole, like the stratigraphic details, are recorded in profiles and sub-surface contour maps, so that then, by combination of the two sets of evaluated data, the tectonic and the geochemical structure of the deposit is clearly apparent.
The tectonic data are especially important in a case where, as a result of secondary actions, changes in the chemical properties of the limestone have occurred on either side of a fault. Although such variations are of a locally limited character, they are liable to cause entirely different raw meal conditions for a time during quarry operation and material processing.

2.1.4.2 Clay component

If the clay component occurs as a solid rock-type material, the requirements applicable to the tectonic investigations are the same as those for limestone.
In deposits consisting of softer material a thorough tectonic investigation is more particularly necessary if adjacent or underlying strata show a distinct deviation from the chemical character of the clay mineral component. Furthermore, water-bearing horizons affected by faults may be encountered during excavation. Also, the stability of slopes is often affected by tectonic conditions, which may give rise to difficulties in excavating the material, especially in countries with heavy rainfall.

2.1.4.3 Overburden investigations

The layer of material which overlies the deposit should be included in the investigation, in order to decide whether such material is to be discarded as useless overburden or can be utilized in the production process, e.g., as part of the clay mineral component or as a sand admixture.
The overburden can be investigated with shallow borings, soundings (penetration testing) or trial trenches.
Sampling is done by the same methods as those for loose rock or soil.
If the overburden is solid rock or similar consolidated material, it is especially important to assess its potential usefulness, for otherwise its removal as mere waste is bound to be a cost-intensive operation (e.g., by blasting).
If the object is only to investigate the depth of overburden, geophysical methods can advantageously be applied. In a case where the overburden is of a loose or fairly soft character, seismic measurements, more particularly by means of the hammer blow technique, are very suitable, as they can be performed quickly and inexpensively. However, this technique does require a relatively level surface of the limestone. If the surface is very irregular, e.g., as a result of underground water percolation, this method of investigation cannot be used. The application of the hammer blow technique in conjunction with penetration tests is especially to be recommended.
With greater overburden thicknesses it is alternatively possible to use a geo-electric method (based on contrasts in the electrical resistivity of strata), which can be very effective more particularly when used in combination with the hammer blow technique.

For interpreting and evaluating the overburden investigations it is most suitable to use a map on which lines of equal overburden depth have been drawn, unless the depth is uniform and very small.

2.1.5 Geophysical investigations

Hammer blow and geo-electric methods represent two simple geophysical techniques which can be used with relatively little effort and expense for determining the depth of overburden, the thickness of consolidated and unconsolidated strata, the detection of waterbearing strata, and ascertaining the ground water table. In addition, determination of the velocity of sound transmission in the ground provides indications as to whether the material can be broken out by ripping.
The hammer blow method is especially suitable in cases where the depth of exploration is limited to 10 – 15 m. The seismic shock (setting up a vibration in the ground) is produced with a heavy hammer which automatically switches on the electronic measuring equipment. A seismic detector (geophone) responds to the ground movements and displays them on an oscillograph. The time it takes for the first shock wave to travel from the hammer to the detector is measured (Fig. 2). If the distance from the hammer to the detector is large enough, the wave produced by the hammer will be refracted at the stratum boundary on penetrating into the underlying material, more particularly the bedrock. The distance between the hammer and the detector is progressively increased, and in each position the wave propagation time is measured.

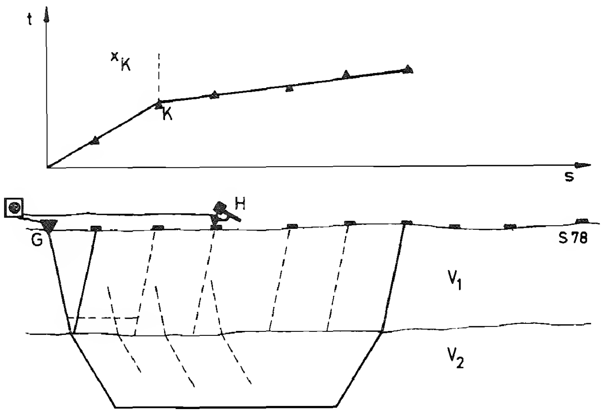

Fig. 2: Propagation and refraction of seismic waves, and time-distance diagram

The results are, to begin with, represented graphically, the propagation time being plotted against the hammer-to-detector distance. The points in the graph are connected to one another by straight lines which show changes in slope according to the number of strata involved. The reciprocals of the slopes of these lines correspond to the wave velocities in the respective strata. The velocities can most quickly be calculated from the linear regression of the measured values, omitting the values close to the "breaks" (changes in direction) because those values are unreliable on account of transition effects:

$y = Bx + A$ where y = time axis (t)
x = distance axis (s)

$$A = \frac{\Sigma t_n \Sigma s_n^2 - \Sigma s_n \Sigma t_n \cdot s_n}{n \Sigma s_n^2 - (\Sigma s_n)^2} \qquad B = \frac{n \Sigma t_n \cdot s_n - \Sigma t_n \cdot \Sigma s_n}{n \Sigma s_n^2 - (\Sigma s_n)^2}$$

$$v_1 = \frac{1}{B}$$ v = velocity in the stratum.

When the lines have been calculated, their intersections can be determined and the distances from these "breaks" on the graph to the origin (point 0) then be worked out. With this distance and the velocities in the two strata it is possible to find the depth at which the interface or boundary surface of the strata is located:

$$D_1 = \frac{x_{k1}}{2} \sqrt{\frac{v_2 - v_1}{v_2 + v_1}} \qquad D_2 = \frac{5}{6} D_1 + \frac{x_{k2}}{2} \sqrt{\frac{v_3 - v_2}{v_3 + v_2}}$$

where D = depth of interface

x_k = distance from "break" to point 0
v_n = velocity in stratum n.

Since this method of seismic exploration operates with only a limited input of energy for producing the ground vibrations, it can be used only for depths not exceeding about 10–15 m and comprising not more than three strata. For greater depths it will be necessary to use explosive charges for producing the vibrations.
The advantage of the hammer blow method is that the equipment with the cables and accessories weighs only about 25 kg and that, operated by one or two men, it is easily possible to measure 10–15 profiles a day. Quite often this method can suitably be used for the mapping of sand or marl horizons or the ground water table in clay deposits.
An important requirement is that the velocities in the respective strata (Table 3) are sufficiently far apart, i.e., differing in magnitude, to enable them to be reliably distinguished from one another.

Table 3: Seismic velocities

residual (weathered) soil	300– 600 m/s
sand, gravel, dry	450– 900 m/s
sand, gravel, wet	600–1500 m/s
clay	750–1500 m/s
shale	1200–2000 m/s
limestone	1600–3000 m/s
sandstone	1600–4000 m/s

Another geophysical method, somewhat more elaborate as regards its application and interpretation, is that of geo-electric exploration, which has a substantially greater range in depth (to about 150–200 m). A distinction is drawn between geo-electric mapping, comprising substantial areas of the subsoil, and soundings which give in-depth information at specific exploration points.
In both cases the so-called four-point arrangement is usually adopted (Fig. 3), comprising an outer pair of electrodes E to which a voltage is applied and an inner pair of electrodes S (probes) across which the resulting voltage is measured.
In the sounding technique, the distance between the electrodes is progressively increased, so that changes in the electric potential distribution in the ground occur and are measured, thus enabling the apparent resistance to be calculated.
The potential distribution in the ground depends substantially on the thickness of the strata with equal electrical resistivity.
If strata differing in their resistivity are present, the pattern of potential distribution at the surface of the ground is altered. The interpretation of the results of the

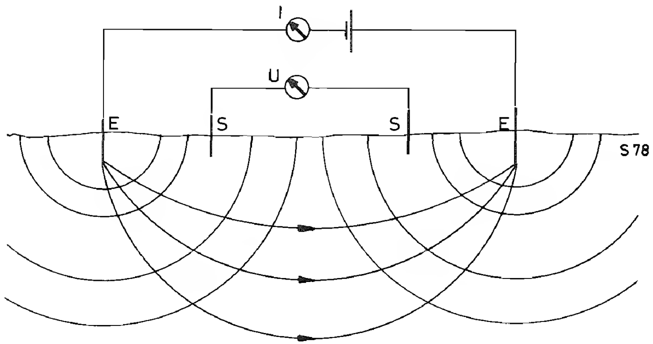

Fig. 3: Current paths and potential distribution in geo-electric measurements (E = electrodes, S = probes)

measurements with progressively increasing electrode distances enables the resistivity and thickness of the individual strata to be determined.
If geo-electric mapping is required, the electrode spacings are kept constant and the whole set-up is moved along to different locations. In this way a map showing lines of equal resistivity is obtained, e.g., enabling large sand inclusions, the surface of a water-bearing stratum or the undersurface of a raw material deposit to be mapped.

2.1.6 Hydrogeological investigations

For planning the quarrying operations it is necessary to know the ground water level on the site to be worked. The most convenient method of obtaining this information is observing the water level in the boreholes. If the water flush technique is used, it is necessary to wait some time until the water introduced into the hole during drilling has dispersed. In any case, the water level observations should be continued over a full year, so as to include seasonal variations.
Hydrogeological observations are liable to be particularly elaborate in limestone deposits with karst characteristics, where a comprehensive network of water level observation points will be needed. If the boreholes fail to provide adequate information on ground water level, geo-electric soundings may be employed, which may moreover be supplemented by geo-electric mapping of the ground water table.

2.2 Laboratory investigations

2.2.1 Chemical investigations

Besides the borings, the chemical investigations associated with an exploration project of the kind described here are responsible for the major part of the expense involved. This being so, it is desirable to use every possible means of working economically by suitably classifying the samples.
For the evaluation of an exploration project for the detection of raw materials for the cement industry it is, as a rule, necessary to know the content of each of the following: SiO_2, Al_2O_3, Fe_2O_3, (TiO_2), CaO, MgO, SO_3, K_2O, Na_2O, Cl and P_2O_5.
Under certain circumstances it will also be necessary to determine the content of organic matter in the limestone and in the clay mineral component, because it tends to undergo oxidation in the preheater and thus, by causing reduction of Fe_2O_3, give rise to incrustations which tend to clog the equipment.
The samples are divided into sections on the basis of macroscopic criteria. There is, however, a risk that variations which may be important in connection with quarry operations planning remain undetected within any particular portion for analysis. For this reason the samples will preferably be subdivided into portions of 1 m length for processing into the actual samples for analysis. For each of these 1 m samples the total carbonate content is first determined, in order thus to obtain information on the variations of the most important constituents, namely, CaO and

MgO in the limestone, SiO_2, Al_2O_3 and Fe_2O_3 in the clay mineral component. In testing the limestone the amount of residue insoluble in HCl should always also be stated, because this residue may contain minerals which significantly affect the MgO content.
After the results for the 1 m portions have been determined, mixtures of the available samples can be prepared, thus providing composite samples comprising several metres of borehole depth. Complete analyses are performed on these. For this purpose it may, to begin with, suffice to perform only a limited number of such analyses for overall guidance,. If these show the alkali content be to be substantially uniform, the alkali analyses may be reduced in number so as to comprise even larger sample quantities, i.e., representative of material from a greater length of borehole. In any case the compounds SiO_2, Al_2O_3, Fe_2O_3, CaO and MgO should be determined only for sample sections of such size that it is possible to alter the quarry operations planning according to the geochemical requirements. For example, if a bench height of 15 m is intended, it is, with sections of 5 m, possible to shift the level of a bench upwards or downwards, in order thus to keep the quarrying geared to, as far as possible, equal geochemical conditions.
X-ray fluorescence analysis has proved very useful for analysing relatively large quantities of limestone and clay samples in a short time. The alkali and the sulphate content will have to be checked by wet chemical analysis, however, because the results of X-ray fluorescence analysis tend to be unreliable except when such analysis is performed by very experienced personnel. Wet analysis will in any case be needed for determining the chloride content.

2.2.2 Mineralogical and petrographic investigations

2.2.2.1 Limestone

In connection with the exploration of limestone for cement manufacture, mineralogical and petrographic investigations have a less important part to play than chemical investigations.
Quite often the limestone occurs in a natural mixture with clay, and in such cases the designation may be based on the chemical analysis, using the nomenclature given by Kühl (1958) (cf. Vol. II, Chapter 2 of his book "Zement-Chemie").
Mineralogical investigations are of interest if the aim is to separate the raw material into lime-rich and clay-mineral-rich components respectively (e.g., for the manufacture of white cement clinker, involving the removal of the constituents containing Fe_2O_3).
Such investigations assume greater importance in dealing with siliceous limestones. For such materials it is necessary to ascertain the distribution of the quartz in the limestone matrix. The type of intergrowth and the grain size of the constituents can be determined in thin sections under the microscope.
The residue insoluble in HCl should also be examined. This can most simply be done by dissolving away the calcareous matter with monochloro acetic acid or formic acid, followed by X-ray examination of the residual material.
Furthermore, the distribution of dolomite can be investigated by means of staining

methods applied to thin sections. However, for practical purposes of assessing raw material deposits it is usually simpler to obtain this information by chemical analysis.
In addition, mineralogical information can be very useful in predicting the severity of wear that will occur in the crushing and grinding machinery.
In many cases the quickest way to obtain adequate information on the mineralogical composition is by X-ray examination of the fine structure of the material.

2.2.2.2 Clay component

Mineralogical and petrographic investigations on the clay mineral component are of interest both in the choice of preparatory processing machinery and in obtaining information on the burning behaviour of the material in the kiln.
In both cases the mineralogical form of the silica, determined by chemical analysis, plays a significant part. Large amounts of free quartz will cause heavy mechanical wear by abrasive action and will, in contrast with the clay minerals, become reactive only at high temperatures.
Swelling clays are liable to cause trouble in storage and in extraction from storage containers or stockpiles.
Information on the mineralogical mode of occurrence of alkalies, sulphates and chlorides can provide clues to possible circulations involving these substances in the cement plant.
These investigations can most simply be carried out by X-ray methods. Alternatively, differential thermal analysis has proved very suitable for the purpose.

2.2.3 Physical investigations

The physical investigations to which the raw materials are subjected usually comprise only the determination of the natural moisture content of the fresh rock and the maximum water absorption.
Grindability and wear tests are performed in connection with the choice and design of the crushing, grinding and other preparatory processing machinery.
In some cases it is also necessary to determine the particle size distribution of clay or sand.

2.3 Evaluation of the results of the investigations

The available results of the investigations should be so processed that all variations in chemical characteristics, workable quantities, materials mixture, and type of machinery to be used in quarrying the deposit can be ascertained from the interpretation and evaluation of the data that emerge.
It is of major importance that the analyses should yield average values for material quantities corresponding to between one and five years' production. Larger quantities may falsify the overall picture, so that useless parts of the deposits may wrongly be rated as useful.

2.3.1 Geochemical evaluation with quarrying operations planning

The first step, in conjunction with planning the quarrying operations, consists in determining the average chemical composition. Then follows the calculation of the raw mix composition. With the results of this calculation the proportion of limestone from the first quarry block required in the mix can be determined. Once this value has been determined, the precise working life of the block can be calculated.
It is possible that the composition of the materials, other than limestone, added to the mix will undergo some change during this period of time, so that a shift in the mix proportions will occur. This must of course be taken into account, so that during the excavation of the first block it may well be that variations in the daily quantities of limestone produced will be necessary.
Similar considerations apply to variations in the composition of the limestone itself. If, for example, a very marly limestone is encountered in a fault zone, it will have to be ascertained how much higher-grade limestone from another part of the quarry will have to be added in order to obtain the required raw mix composition. It may indeed occur that, as a result of such changes in the chemical characteristics of the limestone, the addition of clay to the raw mix can be entirely dispensed with for fairly long intervals. In that case there must of course be sufficient plant available for producing, handling and preparing the extra limestone required. This extra demand for limestone will reduce the working life of the quarry in comparison with the initial estimate.
If, in such cases, operations planning is based on average values over long periods, it may occur that the quarry machinery capacity originally provided will turn out to be inadequate for daily output requirements in course of time. Under such circumstances a crusher, for example, can compensate for this shortfall in capacity only by working longer hours each day.
Such calculations show furthermore that a cement plant which is operated with only two raw material components in the first few years of its working life may, as a result of changes in the average composition of the limestone as quarrying proceeds further into the deposit, require additional corrective components after several years. Alternatively, special arrangements may become necessary such as, for example, the installation of a bypass system to cope with increasing contents of chloride and alkali.
Also, on the basis of such an evaluation of the geological investigations, it is possible to direct the quarrying operations in such a way that certain masses of rock in which some of the constituents exceed the permissible limits can nevertheless be usefully quarried and processed. For example, by varying the floor level of a bench or by working an intermediate bench it may be possible so to control the operations that the limiting concentration is never exceeded.

2.3.2 Calculation and classification of reserves

The information concerning reserves which is contained in the final report of an exploration for raw materials intended for cement manufacture should always relate to workable (recoverable) reserves.

Material excavated for the construction of haulage roads, turning areas, access ramps and safety zones, where no production of rock for processing can be done, should be deducted. Also, some allowance for waste or loss in quarrying should be made.
The total reserve quantity and the working life thereof is obtained simply by adding up the quantities in the respective blocks and the estimated lives of these blocks. Such a calculation should comprise the proved reserves.
The classification procedure for the pit and quarry industry is generally similar to that recommended for ores by the Gesellschaft Deutscher Metallhütten- und Bergleute (Association of German Metallurgical and Mining Engineers, 1981).
"Proved reserves" (category A) comprise reserves which have been the subject of detailed exploration and have been fully investigated with regard to chemical features and their range of variation, bedding, tectonics, preparatory processing, hydrogeological conditions and the legal aspects associated with quarrying the materials concerned. Category B relates to "probable reserves", i.e., the zones which lie adjacent to a deposit containing category A reserves and which have already been explored by borings to such an extent that inferences as to chemical features, bedding conditions and structure, hydrogeological conditions and preparatory processing can be drawn from the experience gained in investigating the category A reserves.
These last-mentioned reserves should be ascertained as the result of the third stage of an exploration project in connection with which the reserves assignable to category B are also estimated.
"Indicated reserves" (category C1) are determinable at the end of the second stage of an exploration project for cement raw materials. These have been investigated on the basis of a network of widely spaced boreholes; the types of rock and their chemical characteristics are substantially known, as are also the structure and bedding conditions in broad outline.
Finally, the "inferred reserves" (category C2) are those which are tentatively determined as the result of the first exploration stage, in which the deposit has been prospected by means of a limited number of individually located boreholes, so that the chemical characteristics and structure of the deposit are known in an approximate and general way.

2.4 Organizing an exploration project

The various activities involved in prospecting for raw materials for the manufacture of cement, as described above, comprise more than just the work of the geologist or geological institution. In order to tackle the task successfully, it is necessary to employ the services of a team of experts from the very outset. It is especially important that this team should include a mining engineer and a process engineer familiar with the cement industry, for only in this way will it be possible to be sure of avoiding serious mistakes which might otherwise be committed already in the planning stage of the exploration project. More particularly, the participation of the process engineer is of major importance in order to ensure that the geochemical investigations are properly geared to the cement industry's needs.

2.5 Using a computer in an exploration project

The evaluation of the geochemical data obtained from the exploration can be substantially speeded up by means of a suitable computation system.
The chemical analyses of the drilled cores can be stored section by section, with associated data relating to the co-ordinates of the borehole, the depth and the thickness of the deposit. By making use of appropriate programs it is moreover possible to store the results obtained from inclined boreholes and from trial pits and, with due regard to the dip of the strata, to obtain a strata-related representation of the geochemical conditions.
Since the benches in the quarry are usually horizontal, the computer can, via the standard deviation, determine coefficients of variation and limiting concentrations for selected areas of the deposit. From this information the bench height and bench sections can then in turn be obtained.
This data collection can be regularly updated and supplemented with further analyses during the subsequent actual quarrying operations, so that predictions of the chemical composition of the material encountered in the individual stages of quarrying can reliably be made.
It is also possible to let the computer produce maps indicating lines of equal chemical concentration, which provide information for determining the direction of quarrying.
Calculations of reserves, evaluations of geophysical investigations and analyses of the bedding conditions can then be carried out.

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II. Quarrying the raw materials

By H. Schuberth

1 Guidelines for quarrying

Raw materials for the cement industry are usually obtained by large-scale open-cast (or open-pit) mining or quarrying operations. Depending on the intended clinker production quantities, quarry outputs may run to several million tonnes of material per year. In order to avoid misdirected capital expenditure it is therefore imperative to obtain reliable information on the raw material deposit, more particularly in terms of quality and quantity. Such information yielded by geological exploration is of decisive importance with regard to the conduct of the quarrying operations. In addition, however, various statutory requirements and obligations have to be fulfilled concerning the excavations themselves, accident prevention and environmental protection. In many cases these so dominate the picture that purely economic and technical considerations of winning the material become secondary to satisfying the statutory conditions.

1.1 Layout of open-cast operations

The most widely used method of quarrying is based on the conventional benching technique, in which the material in the deposit is quarried in several benches ("steps"), one above the other, with predetermined heights of face. If the deposit is located above the level of the cement works, thus involving "hillside quarrying", it is advantageous to use the maximum permissible face heights, because the material broken out of the face falls by gravity to the haulage level, e. g., if large-hole blasting is employed. The restricting conditions on face height may be the accessibility of the top part of the face or the attainable blasthole drilling depth.
Conversely, with "subsurface quarrying", i. e., if the deposit is located below the level of the cement works, it will generally be advantageous to work with relatively low faces, so as to keep to a minimum the expensive work of raising the quarried material from the working floor level to the level of the surrounding ground. The low face is moreover advantageous in cases where quarrying has to be done selectively in order to compensate for variations in the chemical characteristics of the rock. It will then usually be necessary to carry out the quarrying operations in several benches and at several working points simultaneously, so that the composition of the raw material can be controlled. It will only rarely occur that the deposit will consist of material having an ideal composition for cement manufacture, enabling the quarrying operations to be confined to a single face and a single working point. With subsurface quarrying in European latitudes it will usually be necessary to control the inflow of ground water by pumping or other means. The cost of this must not be underestimated.
The various quarry floor or base levels should be connected to one another and to the surrounding general ground level by means of ramps, so that machines, equipment, operating personnel and repair gangs can readily move from one level to another. If the ramps are moreover used as haulage roads for heavy trucks, they should not be more steeply inclined than 1 in 10 and should be sufficiently wide so that two vehicles travelling in opposite directions can conveniently pass each other. Narrower ramps for single-line traffic with passing bays are not to be recommended except perhaps for small quarries with only a few vehicles. The best direction of quarry face advance is along the strike of the bed. In this way it will most easily be possible to meet the safety requirement that hazardous effects of rock pressure or instability must be avoided. If particular reasons necessitate a different direction of face advance, e.g., diagonally inclined, either ascending or descending, the danger of falling rock from overhanging parts should be counteracted by increasing the batter of the working faces. It should also be borne in mind that surface water is liable to collect on, and run off along, such bedding or parting planes, thus forming a possible cause of rock slips.
The height of the working face is, for example in the Federal Republic of Germany, subject to statutory regulations with regard to permissible maximum values depending on the method of quarrying or the size of machines used. The slope and width of the benches should be suited to the nature and stability of the rock and to the method of quarrying.

1.2 Quarry equipment

The mechanical equipment of the quarry, more particularly the number and size of the machines, will depend on the intended rate of production and on the haulage distance. With regard to the economy of the operations it can, roughly speaking, be said to improve with increasing size of the machines employed, provided that a sufficiently high rate of production in the quarry will enable a correspondingly high degree of plant utilization to be achieved. In many cases, however, fulfilment of this requirement is restricted by quality considerations, more particularly when a certain constant average quality of the output from the quarry has to be obtained by the controlled combining of various grades of rock.
Of especial importance is the proper interadjustment of the machines employed, i.e., ensuring that they are duly suited to function efficiently with one another, more particularly in the operations of loading, haulage and crushing.
Thus, the loading machine should be so suited to the haulage trucks, and vice versa, that the number of loading bucket operating cycles for filling a truck is

between three and eight, the larger number being applicable to the smaller bucket. From the economic point of view it is important not to allow the capital tied up in the engines and running gear of the vehicles to remain idle for too long periods. They must earn their keep!

On the other hand, the receiving capacity of the crusher should be large enough to accept the full contents of a haulage truck discharged in just one dumping (or tipping) operation. Finally, the size of the rock pile fragments fed to the crusher should not be so large as to cause jamming in the feed opening.

In planning the quarry, the need for providing intermediate storage directly before or after the primary crusher should be considered. Such buffer capacity makes the rate of quarrying to some extent independent of the rate of further processing and can thus be invaluable in maintaining continuity of supply in the event of temporary hold-ups in quarrying activities (see also Chapter B. III).

2 Overburden

It will only seldom occur that a raw material deposit is not covered by a layer of overburden or that the overburden can be directly excavated and processed along with the actual deposit because the chemical composition fits in with that of the raw mix itself. In any case the overburden will have to be removed separately from the material of the deposit. It will either have to be dumped as unprocessable material (along with any unwanted inclusions and impurities from the deposit itself) or be suitably stockpiled, so that it can be reclaimed in controlled quantities and mixed in the right proportion with the main material from the deposit.

2.1 Overburden removal

The method of removal will depend on the following factors relating to the overburden:

- strength and hardness; soil or solid rock;
- thickness of the layer;
- haulage distance;
- loadbearing capacity;
- susceptibility to weathering.

Provided that rock overburden can be suitably broken up by drilling and blasting or by ripping, the following conventional types of machine can be used for its removal:

- backacting excavator (back-hoe);
- dragline excavator;
- bulldozer.

In general, the ground surface which is as yet intact will, on account of its vegetation, have better bearing capacity for loads than ground that has already had its top layer removed. As indicated, the preferred machines for topsoil digging – nowadays mostly with hydraulic controls – are the backacter and the dragline.

The backacter is better able to remove unconsolidated material from any fissures, crevices or dolines (swallow-holes). On the other hand, the dragline has a larger outreach and greater digging depth. Besides, the dragline bucket, suspended loose from its rope, can swerve to miss obstacles on a rough rocky surface, so that the excavator is not subjected to excessive wear and tear. If the material to be handled is fragmented rock, the pieces will have to be fairly small, however.

With both types of excavator it is necessary to use some form of haulage machine for removing the excavated overburden material. In most cases, various types of truck are used for such purposes. Multi-axle articulated dump trucks with multi-wheel drive have been found most suitable because of their good manoeuvrability on the generally bad ground on which they have to travel. Alternatively, the excavated material can be loaded, via suitable feed devices, onto belt conveyors in cases where these can be economically used in order to cope with large handling quantities or to meet other requirements.

The bulldozer can suitably be used as a means of overburden stripping if the handling distances are not too great, if there is only a limited thickness of overburden or if highly cohesive soil leaves no alternative to this method without necessitating extensive additional measures (construction of roads). Furthermore, a bulldozer is usually a very useful piece of equipment for work on building up the soil tips.

Besides the above-mentioned "classic" overburden handling machines, other types of machine are used for special purposes or under special conditions, such as face shovels, scrapers, scraper-dozers, wheel loaders, crawler loaders, possibly even bucket ladder excavators or small bucket wheel excavators.

2.2 Storage of overburden material

The planning of suitable piles or tips for dumping the overburden material, more particularly with regard to quantities to be stored and favourable location relative to the source of the material – and, of course, outside or at the edge of the deposit to be quarried –, should be done with considerable care. It often occurs that, due to inefficient planning, the area reserved for overburden dumping turns out to be inadequate and can subsequently be extended only at considerable expense or indeed not at all. As for the technical layout of an overburden pile the following points call for consideration:

The pile should be well and firmly based on the subsoil. If the latter is waterlogged, it should be drained. The overburden material should be placed layer by layer, for only in this way will there be adequate compaction of the dumped material by the haulage and handling vehicles travelling over it during the build-up of the pile. The layers should not exceed 8 m in thickness. Each individual layer should end at a distance of 4 m before the one below, so that a berm is formed. The berms should be inclined slightly backward, and surface water run-off should be intercepted in adequate discharge channels and removed under controlled conditions, in such a way as to prevent erosion on the berms and slopes or at the toe of the overburden pile.

Slopes should never be steeper than 1 : 2 and should be grassed and planted as soon as possible after being given a covering of topsoil, so that the vegetation can help to keep the soil in position and scouring action by rainwater is avoided. The build-up of an overburden pile should be so controlled in terms of time that it will not have to go through the winter months, with heavy rain and/or snow, while its slopes remain devoid of vegetation because grassing them was left too late for the grass seed to germinate. In addition, an intercepting ditch should be dug at the toe of the pile. Any material washed down can settle in this ditch, and excess rainwater collecting in it can be discharged under controlled conditions after sedimentation of solids.

3 Breaking out the rock

3.1 Drilling and blasting

Drilling and blasting continue to be the favoured combination for breaking out the material, i. e., dislodging it from the quarry face and fragmenting it. Although it has, in recent years, increasingly been brought into discredit on account of the noise and vibrations that unavoidably arise and has, as a result of environmentalist activity or statutory regulations, often been restricted and sometimes indeed banned, the real economic advantages it offers in most cases are still utilized wherever the opportunity exists. In addition, efforts are continually being made, and with some success, to adapt the drilling and blasting technique to the specific conditions of the deposit and the local environment and thus reduce its undesirable effects to a minimum. Even so, it must be remembered that the steady growth of "environment-consciousness", both on the part of the authorities and of the general public, often rules out a choice of quarrying methods based on purely economic considerations. In such cases a different method of breaking out the material will have to be applied, such as ripping or stripping.

3.1.1 Drilling large-diameter holes

The large-hole blasting method (sometimes called well-drill blasting) is now predominant in quarrying in open-pit workings. It can bring down large masses of rock from the face, suitably fragmented for loading, with due regard to the layout of the quarry and the planned progress of operations, while avoiding severe ground vibrations and involving only a small amount of secondary blasting for breaking up over-large fragments.

The economic advantage of large-hole blasting, and therefore its widespread use, are due to the fact that the operations of "drilling" and "loading of the rock pile" can be carried out quite independently of each other.

The definition of large-diameter blastholes is, in Germany, linked to the relevant accident prevention regulations and relates to holes more than 12 m in depth. Irrespective of this statutory definition, the engineer on the job rates any hole exceeding 50 – 60 mm diameter as coming within this category. The predominant

diameter range in current German use is between 60 mm and 105 mm, occasionally up to 150 mm. In other countries, more particularly in the USA, larger diameters are preferred, namely, 225 – 300 mm and even more. In densely populated areas the acceptable blasthole diameter is often limited by considerations of ground vibrations, which are liable to be excessively severe if the charge fired per hole or per stage of detonation is too large.

3.1.1.1 Single-row blasting

In most cases the large-diameter blastholes are drilled in one row parallel to the slope of the quarry face. The most favourable slope is between 70° and 80°. In order to ensure proper break-out of the toe of the face, the holes are usually drilled so as to extend a certain a short distance below the level of the quarry floor (sub-drilling). With face heights commonly around 20 m, a sub-drilling depth of about 1 m has become the established practice. It should be noted, however, that particularly the explosive charge in the sub-drilled part of the holes is likely to cause the most powerful ground vibrations. In Germany, face heights in excess of 30 m are now prohibited on account of the accident hazard associated with them. The great majority of faces in quarries are about 20 m in height or less. There is a trend towards reducing the height because this makes for better selectivity in conducting the quarrying operations.

There is a whole range of possible variations in large-hole blasting practice, from single-row and multiple-row blasting with or without toe holes to so-called surface blasting.

The choice of blasting method, more particularly the number of blasthole rows, depends on the properties of the rock as well as on the vibration effects that can be tolerated. For example, holes disposed in a number of rows over a certain area are more likely to offer a suitable solution in brittle easy-to-shatter rock than in tough rock fracturing into large blocks.

The column of explosive in a blasthole should, if possible, extend continuously from the bottom of the hole up to the stemming. Only in this way can the cost of producing such large blasthole volumes be fully utilized by working with sufficiently large hole spacings and burdens.

It often occurs, however, especially in heavily fissured rock, that the blasting energy is insufficient to dislodge the more heavily restrained rock mass at the toe. But if the geometric features of the blastholes (diameter, burden, spacing) are sufficiently reduced to ensure break-out of the toe, it will frequently be necessary to use intermediate stemming in the upper part of the holes in order to avoid wasteful use of explosive and the risk of large rock fragments being hurled out with dangerous force, particularly in places where irregular break-out at the quarry face has locally reduced the burden. In such cases the waste of a certain proportion of expensively drilled blasthole volume will be unavoidable.

These drawbacks may be overcome by suitably increasing the blasthole volume at the toe of the face, so as to obtain a larger quantity of explosive charge where it is needed most.

This is usually done by systematically drilling so-called toe holes from the quarry

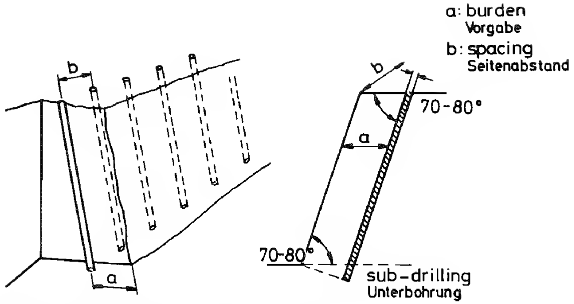
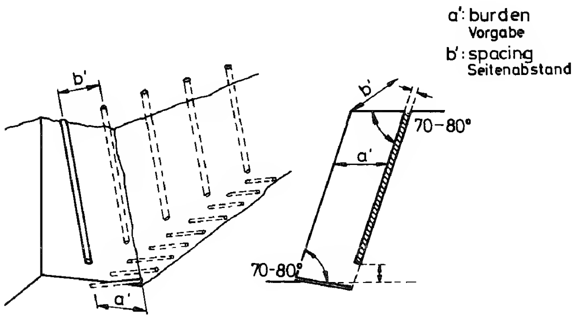


Fig. 1: Blasting with large-diameter holes



Blasting with large-diameter holes and toe holes

floor, these being of such diameter and spacing as to achieve the required extra blasting effect at the toe. With the right type of drilling machine and the introduction of free-flowing granular explosive into the toe holes by means of blowing equipment, this procedure may, in suitable rock, be more economical than having subsequently to carry out supplementary drilling and blasting to dislodge those portions of the toe which have remained standing after the firing of the main charge.

All the same, the techniques for obtaining greater blasthole volume at the toe of the face, though offering the advantages mentioned here, are not in very widespread use. The reason probably lies in the difference in technical development of the machines for drilling vertical and those for drilling horizontal holes, in the relatively low cost of the ANC (ammonium nitrate-carbon) explosives chiefly used in vertical holes, and the need to remove all rock pile (fragmented rock) from the toe of the face before toe hole drilling can commence.

3.1.1.2 Surface blasting

On account of the above-mentioned drawbacks, so-called "surface" blasting is gaining wider acceptance. With this technique the rock is loosened in consequence of the fragmenting effect of blasting in a number of holes distributed over a certain area instead of being disposed in one row. This method is especially suitable for selective quarrying or when separate loading of different materials found in the same quarry is required, since the location of the material remains substantially unchanged after blasting. There is essentially a lifting action and bulking of the rock as a result of fragmentation. A drawback is that this method requires about twice as much drilling (in terms of hole length) and twice as much explosive. The holes themselves are generally of much less depth than those in conventional large-hole blasting from a face. An advantage of surface blasting is that the amount of subsidiary work – such as secondary fragmentation, quarry face trimming and floor levelling – is generally less.

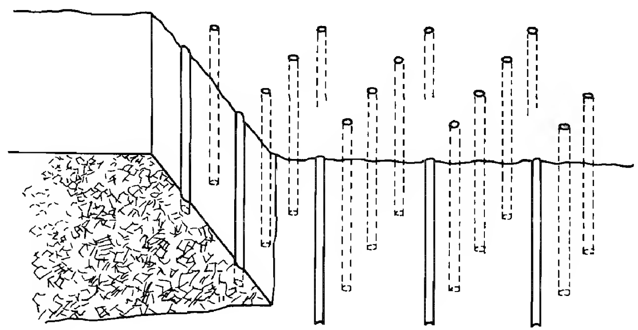


Fig. 2. Surface blasting with large-diameter holes

3.1.1.3 Drilling tools

Rotary drilling and percussive rotary drilling are almost the only methods used for forming the blastholes in quarries for cement raw materials. The drilling tool is generally a step bit; for larger diameters a roller bit is sometimes used or, in percussive rotary drilling, a cross bit or a stud bit. The last-mentioned type of bit is claimed to be especially advantageous in hard rock because of the higher specific feed pressure that can be applied. Besides, it is better able to cope with fissured rock because it cannot jam so easily in the crevices. With down-hole hammers the drilling force is developed at the bottom of the hole instead of being transmitted down through the drill rods, so that the latter are less severely stressed, while the drilling machine itself is also relieved of mechanical load. Besides, there is less likelihood of deviation of the drill hole from the vertical.

3.1.1.4 Drilling machines

Modern rotary drilling machines are operated by just one man. They mostly have fully hydraulic drive systems, are reliable in operation and attain drilling rates of up to 30 m/hour, depending on the nature of the rock and the diameter of the hole. The power pack, compressor, hydraulic units, drilling mast, rod magazine, operator's platform and dust suppression system are mounted on a traction unit usually equipped with crawler tracks. The prime mover is generally a diesel engine. Although it is more expensive in energy consumption than an electric motor, it is nevertheless preferred because it provides better mobility of the drilling machine and makes it independent of power feed cables. On some machines a slewing ring enables the superstructure to swivel on the crawler chassis, thus enabling unproductive manoeuvring of the whole machine to be reduced. The use of increasingly long drill rods likewise aims at increasing the efficiency of the machine, a trend which has led to the development of the "single-pass" machine which drills the hole to its full depth with just one long rod, i.e., without having to couple successive rods as drilling proceeds. Rubber-tyred traction units can suitably be used under circumstances where the machines each have to operate at a number of different points, or at different working levels or indeed in different quarries, so that substantial distances have to be travelled. However, the ground on which they travel will have to be of sufficient bearing capacity to carry their weight.
There is no doubt that the fully automatic one-man-operated rotary drill requires more skill on the part of the operator, and also more servicing, than does the percussive rotary drill powered with compressed air and mounted on crawler tracks. These machines are of relatively low weight. With a suitable compressor in tow, a machine of this kind can move about under its own power even on difficult terrain. The drill guide mast can be tilted and swivelled in all directions, so that a wide variety of drilling duties can be performed. These machines are the preferred type in small and medium-size quarries and in cases where highly skilled operating personnel are unavailable.

3.1.2 Blasting

When the blastholes have been drilled, they are charged with explosive and the charges are fired. The object of blasting is to loosen and fragment the rock so as to obtain a rock pile suitable for loading. The amount of explosive to be used in any given case will depend on the specific explosive consumption, i.e., the amount needed for producing a tonne of rock pile or for loosening and fragmenting a cubic metre of solid rock. It is an empirical value which varies from one set of quarrying conditions to another and should be known in any quarry where production is in progress. When opening up the quarry, this value can be determined by reduced-scale trial blasts based initially on known average values from practical experience under comparable conditions. The specific explosive consumption is mostly between 200 and 400 g per m^3 of solid rock. It does, however, vary within wide limits, depending to a great extent on the nature of the rock – whether it is hard,

soft, compact, fissured or affected in some other way. The blasthole location grid, i.e., the spacing and burden dimensions, is determined on the basis of the calculated quantity of explosive needed for breaking out the intended quantity of rock by blasting with holes of given diameter (which in turn depends on the type of drilling machine available). The appropriate relationship of blasthole spacing and burden can be expressed as a product of these two dimensions (in m^2). The value of this product in any given case can be calculated by determining the quantity of explosive (in kg) which can on average be charged per blasthole and dividing this quantity by the required specific explosive consumption (in kg/m^3 of rock).
The smaller the spacing and the burden, with correspondingly smaller blasthole diameter, the better will be the fragmentation obtained, because the explosive will be more uniformly distributed along the face. A finer location grid is more particularly advantageous in dealing with thick-bedded rock tending to produce a coarsely fragmented rock pile.

Table 1: Single-row blasting with large-diameter holes

hole diameter (mm)	blasthole grid (m^2) corresponding to m^3 of rock per drilled metre	burden (m)	spacing (m)	blasthole volume (litres/drilled metre)
92	20	3.5 to 5	3.5 to 4.5	6.6
105	24	3.5 to 6	3.5 to 5	8.6
150 without toe holes	30	4 to 7	4 to 6	17.6
150 with toe holes 76 mm ∅	50	5 to 10	5 to 7	17.6 + 4.5
225	50	6 to 10	5 to 8	29.8

With increasing blasthole diameter, spacing and burden there is an increase both in the proportion of very finely fragmented material (due to shattering of the rock in the immediate vicinity of the charge) and in that of large lumps (dislodged from the parts of the rock farthest from the charge). A coarse grid of this kind will as a rule be economically advantageous only in rock which is fractured, finely fissured and brittle. Blasting by the tunnelling method, now seldom used, represents an extreme case of firing large concentrated charges.
As already stated, the aim is to fill the entire blasthole with explosive, if possible.

The stemming inserted in the top part should as a rule have a depth equal to the burden.

A whole range of explosives is available to the quarry engineer. It extends from powder to gelatinous explosives and includes slurry explosives; there are low explosives and high explosives, as well as intermediate types; explosives which are used in cartridge form and those which are used in bulk. Careful consideration of the choice of explosive, so that the most suitable type for the job is used, makes for greater economy.

In raw materials quarrying for the cement industry there is a trend towards the preferred use of the inexpensive ANC (ammonium nitrate-carbon) explosives. These are superseding the more expensive gelatinous types (gelatines, gelignites), whose use is now mainly confined to that of a priming charge for initiating the slower ANC, but even here they are making way for the heavier-grade detonating fuse (more particularly the 40 g fuse). They are, however, still in common use for secondary blasting, i.e., the further reduction of oversize fragments of rock by individual drilling and blasting. In quarries where ANC explosives are used, their proportion is seldom below 70% of the total explosive consumption. In certain cases it may be as high as 99% or more.

Another reason why the ANC explosives (known also as ANFO = ammonium nitrate-fuel oil) have rapidly been gaining ground is their high degree of handling safety and the possibility of conveying them in special mixing/loading trucks to the actual site of blasting – provided that sufficiently large quantities of explosives are consumed to make this economically attractive and that the quarry floor offers a reasonably level riding surface. The actual explosive mixture of ammonium nitrate and diesel oil is produced "on the spot" in the truck and is pumped through a hose into the blastholes. Alternatively, it can be introduced into the holes with special pneumatic loading devices. The wage costs involved in loading the blasting charges by this method are very low. Besides, there are substantial savings due to eliminating the transport of the explosives from the magazine to the actual site of blasting and dispensing with any handling of explosives at the magazine itself. It should not go without mention, however, that the economically advantageous method of "on the spot" bulk delivery of explosives by truck to the blastholes is used also with other types of explosive, more particularly the slurries.

The degree of filling and therefore the charge efficiency of the blastholes depends not only on the density of the explosive itself, but also on the blasthole and cartridge diameters employed or on whether the explosive is used in bulk form. This should be duly borne in mind when calculating the explosive quantity in kg per linear metre of blasthole. The degree of filling is around 70% for powder explosives in cartridge form; around 90% in the case of slurry or gelatine-type explosives; and around 100% when explosives are used in bulk.

For reasons of safety, blasting charges should be fired only by electric detonation. Detonators (blasting caps) with varying degrees of sensitivity are available for the purpose. They are produced as instantaneous detonators or delay detonators, the latter being of the ordinary delay (usually half a second) or the millisecond delay type. The last-mentioned detonators are manufactured in a range of delay periods differing by some tens of milliseconds. The detonating current is now usually

Table 2: Charge in kg per lin. metre of blasthole in relation to degree of filling and diameter of holes, for explosives of various densities

degree of filling	mm blasthole diameter														
	40	50	60	65	70	75	80	85	90	95	100	105	110	120	150
	kg ANC explosive, in bulk, density approx. 0.9														
100	1.1	1.8	2.5	3.0	3.5	4.0	4.5	5.1	5.7	6.4	7.1	7.8	8.6	10.2	15.9
	kg ammonium nitrate explosive, ANC explosive in cartridge form, density approx. 1.0														
100	1.3	2.0	2.8	3.3	3.9	4.4	5.0	5.7	6.4	7.1	7.9	8.7	9.5	11.3	17.3
90	1.1	1.8	2.5	3.0	3.5	4.0	4.5	5.1	5.7	6.4	7.1	7.8	8.6	10.2	15.9
80	1.0	1.6	2.3	2.7	3.1	3.5	4.0	4.5	5.1	5.7	6.3	6.9	7.6	9.0	14.1
70	0.9	1.4	2.0	2.3	2.7	3.1	3.5	4.0	4.5	5.0	5.5	6.1	6.7	7.9	12.4
60	0.8	1.2	1.7	2.0	2.3	2.7	3.0	3.4	3.8	4.3	4.7	5.2	5.7	6.8	10.6
	kg gelatinous ammonium nitrate explosive, ** slurry in cartridge form, density approx. 1.5														
100	1.9	3.0	4.2	5.0	5.8	6.6	7.5	8.5	9.5	10.6	11.8	13.0	14.3	17.0	26.5
90**	1.7	2.7	3.8	4.5	5.2	6.0	6.8	7.7	8.6	9.6	10.6	11.7	12.8	15.3	23.9
80	1.5	2.4	3.4	4.0	4.6	5.3	6.0	6.8	7.6	8.5	9.4	10.4	11.4	13.6	21.2
70*	1.3	2.1	3.0	3.5	4.0	4.6	5.3	6.0	6.7	7.4	8.2	9.1	10.0	11.9	18.6
60	1.1	1.8	2.5	3.0	3.5	4.0	4.5	5.1	5.7	6.4	7.1	7.8	8.6	10.2	15.2
	40	50	60	65	70	75	80	85	90	95	100	105	110	120	150
	mm blasthole diameter														

supplied by a condenser discharge blasting machine (exploder). This is a reliable type of machine which is increasingly superseding the earlier electrodynamic exploder with direct discharge of current. For blastholes exceeding 12 m in depth the use of detonating fuse is compulsory under German regulations. In such cases the detonators are fitted to the end of the fuse outside the hole. If the relevant regulations allow the electric detonator to be used for firing a primer cartridge at the foot of the hole, detonation will be initiated in the region where the highest degree of restraint from the surrounding rock exists, so that then the greatest blasting effect will be obtained. Besides, the detonation report will be more muffled and thus cause less nuisance to neighbouring residents.
It should be mentioned, however, that the older method of blasting with safety fuse and appropriate detonators is still used to some extent. This type of fuse consists of a train of black powder enclosed in a waterproof tubular casing and has to be lit.

3.1.3 Cost

It is not possible to give generally-valid information on the cost of blasting with large-diameter holes. It will depend on a variety of determining factors, including: the type and stability of the rock, the size and utilization of the drilling machine, the type and method of use of the explosive, requirements as to the fragmentation of the rock pile in connection with available loading or further processing facilities, etc. In approximate terms it can be stated, however, that the specific cost of drilling and blasting per tonne of loadable rock pile shows a slight hyperbolically decreasing trend, so long as the diameter remains within reasonable limits, as envisaged in the foregoing description of the blasting operations.
However, in seeking to take advantage of this trend it will often occur that economic limits are encountered, more particularly when the drilling machine capacity substantially exceeds the quantities of rock pile actually needed by the cement works in a given period. In such cases it often works out cheaper to let an outside firm carry out the entire drilling and blasting operations on a contract basis. Obviously, this is more likely to be an attractive solution where relatively small quantities of material are required than in medium-sized or large quarries, though local conditions and other considerations will of course play a part.

3.1.4 Tunnelling method

In the tunnelling method of blasting (known also as "coyote blasting") fairly large charges are fired in tunnels driven into the face. It is now hardly every used. It could, however, be considered in cases where capital expenditure has to be kept low or where the surface of the raw material deposit is inaccessible to drilling machines, e. g., in very rough or mountainous country. The major drawbacks of this method are the tunnelling work itself, the severe vibrations set up by the blasts, and the very coarse fragmentation achieved, necessitating much secondary blasting.

3.1.5 Series firing of small-diameter blastholes

This technique is still used where relatively small quantities have to be fragmented, e. g., in dealing with residual rock masses, or in a supplementary capacity to other blasting methods for dealing with particular features of the deposit. The holes, of small diameter, may be drilled horizontally into the face or vertically or at any angle. They may be located side by side or one above the other; they may be parallel to one another or fan out. If the burden is kept small, the explosive consumption is often quite low and fragmentation is good, i. e., relatively few large fragments requiring secondary blasting are formed. This result will usually depend on achieving a uniform distribution of the explosive charges within the rock mass to be broken out by drilling a large number of holes carefully suited to the bedding conditions. A major drawback is that this method is very labour-intensive, especially if separate drilling platforms have to be erected against the quarry face. It also involves by no means negligible accident hazards because the men have to work close up against the face and spend fairly long times there.

3.1.6 Secondary blasting

No blasting method can completely avoid the production of a certain proportion of oversize pieces of rock ("boulders"), though it is often possible to keep this proportion down to a minimum by suitable choice of blasting method. These oversize pieces have to be further reduced, otherwise they would have an obstructive effect on the further operations of loading, haulage and crushing. The maximum size of boulders that can be tolerated will of course depend also on the size and capacity of the handling and crushing plant used in the quarry.
Boulders are usually broken up by blasting ("secondary blasting") because this nearly always gives a suitably fragmented product, whatever the type of rock. This is mostly done by drilling small-diameter holes to a depth equal to a little more than the diameter of the boulder. They are charged with 60 – 90 g of explosive per m^3 of rock, stemmed and detonated (electrically, if possible).
Another method of secondary blasting is called "mudcapping" or "plaster shooting". In this case a substantially larger quantity of a gelatine-type high explosive, characterized by high detonation velocity, is used (250 – 500 g/m^3). It is simply applied to the surface of the boulder, well stemmed and detonated. The drawbacks of this technique are that it is very noisy (environmental nuisance) and often not economical either, so that it is tending to go out of use.
Secondary fragmentation by mechanical methods in lieu of blasting is gaining ground. They are based either on the pounding action of a heavy dropweight or on demolition of the boulder with pneumatic or hydraulic breaking hammers. Obviously, the success of such methods will depend to a great extent on the hardness and toughness of the rock, the underlying material on which the boulder rests, and the size and power of the mechanical equipment employed. This being so, it is necessary to carry out tests to find out if mechanical secondary fragmentation is economical before a decision is made. Another drawback is that, with such methods, it often occurs that some of the secondary fragments are still really too large.

3.1.7 Storage of explosives

The primary consideration with regard to the storage of explosives is that of safety. Hence it is can be presumed that in no country anywhere in the world the accumulation of storage of stocks of explosives can be permitted without any restriction by official regulations of some kind. In the Federal Republic of Germany the statutory requirements are laid down in the second Decree for the implementation of the Explosives Act (of 23.11.1977), including the Appendix containing the principal technical regulations, and furthermore in the relevant Guidelines in which these requirements are further elaborated.
All these regulations are directly applicable, i.e., they do not become effective only after the granting of a licence to store explosives. Since the whole question of storage involves some legal complexities, it is advisable to seek guidance for the relevant inspection authorities at the very outset, when the setting-up of an explosive magazine is contemplated. This precaution can save a lot of frustration, time and money.
Although these statutory requirements are applicable only to Germany, it can be helpful to seek guidance from them on the safe storage of explosive in countries where these matters are not subject to such close regulations.
Of major importance is the classification into "storage categories" to which potentially hazardous explosive materials are assigned. For the present purpose only category 1.1 is of interest, comprising the industrial explosives and black powders.
The safe distances from the explosives magazine to residental areas and public highways, depending on the quantity of explosive stored, are stated in Supplement 1 to the Appendix. These distances can permissibly be varied within certain limits, depending on the importance of the areas or installations to be protected and on the constructional features of the magazine.
The general requirements applicable to explosives storage are laid down in the second part of the Appendix. The most important of these, besides the safe distances, is a general prohibition on the storage of these materials in the open air or in vehicles. It is also stated that no explosives are allowed to be stored directly at access ways to places of work. Emphasis is laid on fire protection arrangements, and precautions against the action of electricity and against theft or unauthorized removal of explosives are outlined, e.g., a ban on windows, the requirement that suitably strong doors, walls and roofs be provided, and that the magazine be reliably locked up and the keys kept in safe custody. The safety precautions should – in view of the fact that the manufacture of "home-made" explosives by criminals and terrorists is now commonplace – concentrate more particularly on detonating equipment such as detonators, detonating fuse and electric exploders.
Other regulations are concerned more particularly with the construction, fitting-up and operation of the magazines.
In the fourth part of the Appendix the requirements applicable to the storage of explosive materials outside a magazine are outlined. These comprise what are defined as small quantities needed for day-to-day use in the quarry and held readily available at various conveniently located points. Also included is the mobile storage of such quantities in containers, cabinets or site vehicles.

3.2 Ripping

Another method of breaking out the rock, as an alternative to drilling and blasting, is represented by ripping. A distinction is to be drawn between the ripping of rock from horizontal surfaces and ripping from vertical faces.
Obviously, the ease or difficulty with which any particular type of rock can be dislodged and fragmented by ripping will depend to a great extent on its hardness and compactness, as well as other geological and tectonic properties. Factors that make for easier ripping are heavy fissuring of the rock, thin but well developed bedding, coarsely crystalline structure, inhomogeneity, zones affected by weathering or tectonic action. Conversely, homogeneous, solid, fine-grained rock without weak spots is difficult to break up by ripping.
"Horizontal" surface ripping, which is the commoner method, is carried out with the aid of one or more ripping teeth mounted at the rear end of a heavy crawler tractor. The teeth penetrate into the rock and drag grooves or furrows in it as the tractor travels. The material loosened in this way is then shifted by bulldozing.
The most reliable way to decide whether a particular type of rock is indeed suitably "rippable" is by practical trials.
A simpler, though not nearly so informative, method is based on the principle of seismic refraction. The transit times of shock waves in the subsoil are measured, these waves being produced by hammer blows applied to steel plates located at varying distances and being detected by a seismic pick-up device (geophone).
The velocity of propagation of these waves in the rock is a measure of its in-situ strength and thus provides an indication of the probable rippability. The relationship between wave velocity and ripping characteristics has been determined empirically from numerous observations. Though of course the power and weight of the machines concerned are major factors, it can broadly be stated that with the crawler rippers in present-day current use the types of rock which are of interest to cement manufacture, such as limestone or shale, are likely to be suitably rippable if the seismic wave velocity does not exceed about 2000 – 2500 m/second.
As Fig. 3 shows, only the latest super-heavy crawler rippers can tackle rock in which this limit is somewhat exceeded. However, the seismic wave velocity can offer no more than approximate guidance. Determining the rippability and the ripping effort of rock is still more of an art than a science. It requires much experience to hit upon the optimum combination of ripping speed, depth and spacing of the furrows.
The design, number and method of attachment of the ripping teeth are of major importance in connection with this. The teeth come in various shapes, straight or curved, each type being more particularly suited for certain types of rock. The design of the tooth tip also plays a part. Thus, short tips are better suited for rock which is difficult to penetrate, whereas long ones are more effective in abrasive rock. Medium-length tips set to the correct cutting angle can develop high breaking-out forces and can cope adequately even with rock of an abrasive character. If the ripping attachment is mounted so that it can swivel about a point of rotation, the cutting angle of the teeth will vary with their depth of penetration. This

method of mounting is generally restricted to certain types of rock. For many other types the parallel-motion system of mounting is more suitable, because here the optimum cutting angle, once it has been correctly set, remains unchanged irrespective of the working depth. There is, finally, a combination of these two systems in that the working angle is adjustable, usually by means of a hydraulic ram, the reason being that the optimum angle for penetrating into the rock at the start of work may differ from the optimum angle for the actual ripping operation itself. As a rule, a single tooth should initially be tried. Only in relatively easily rippable rock will it be possible to operate with several teeth.

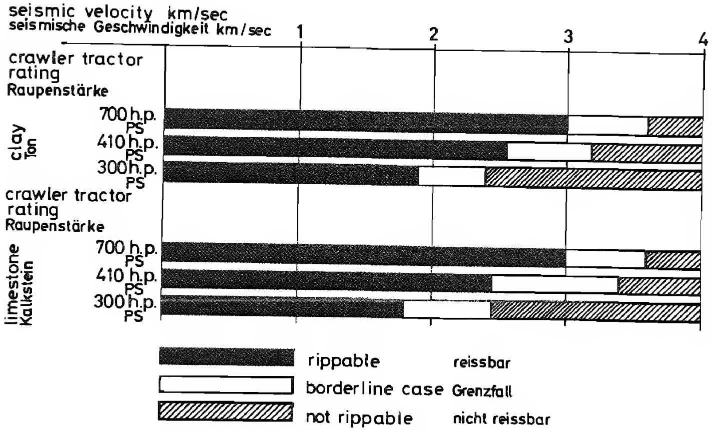


Fig. 3: Ripping capacity of crawler rippers

The most economical quarrying technique in given circumstances will have to be determined by trials. For instance, the spacing of the ripping furrows will affect the resulting fragmentation of the material. The maximum attainable penetration depth is not necessarily always the optimum. On sloping ground the ripping direction most often employed is downhill; this is likely to achieve the highest output of loadable rock despite idle uphill travel of the ripper. Failure to remove all the dislodged material will cause "cushioning" of the tractor on its next ripping pass and will increase the friction factor between the crawler tracks and the solid rock underneath. Sometimes it may be advantageous to do occasional blasting with light charges in cases where intermediate strata of rock resistant to ripping are encountered.

The ripping method of breaking-out in quarries must be judged in comparison with the alternative of drilling and blasting. Ripping may be preferable in one or more of the following cases:

- the effect of ground vibrations due to blasting presents an environmental problem and thus seriously restricts the operations in the quarry (though it should be borne in mind that ripping may introduce its own problems, more particularly due to noise and dust emission);
- the quarrying of the material over large areas by ripping achieves an advantageous degree of homogenization in deposits of inhomogeneous composition;
- residual areas of the workable deposit, which have been left standing because their proximity to vulnerable installations ruled out blasting (e.g., near roads, railways, buildings), have to be quarried as well.

Ripping requires large working areas and extensive opening-up of the quarry. A drawback is that, depending on seasonal factors, the raw material quarried by this method will absorb up to 2% more moisture in the quarry, and this extra moisture will of course have to be removed in the cement works, involving correspondingly higher energy input.

The performance and therefore the cost of ripping depend very much on the length of the ripping passes and bulldozing distances. As a rule, shorter passes are more advantageous. With passes up to about 50–60 m in length, which are to be regarded as the maximum, outputs (production rates) of up to 550 t/hour can reasonably be expected when the usual heavy crawler rippers of up to 60 t overall weight and up to 500 h.p. engine ratings are employed.

Ripping makes severe demands on the robustness of the machines. The frame and undercarriage have to be very stable, for the ripping action develops not only high peak values of the traction force, but also swerving moments that tend to push the crawler tracks off course. High operating, maintenance and repair costs have hitherto generally made ripping unattractive as an alternative to quarrying by drilling and blasting, except in cases where there are compelling reasons not to employ the latter method, more particularly in cases where environmental protective restrictions have to be complied with. Hence the development of ripping will continue to be watched with interest. The operating results obtained with the super-heavy crawler rippers of about 86 t overall weight and 700 h.p. engine rating, which have latterly appeared on the market, will have to be awaited before a more definite assessment can be made. It may then well be that, with sufficiently high levels of plant utilization, ripping will offer an economically more acceptable alternative to drilling and blasting.

3.3 Stripping

Stripping with bucket wheel or bucket chain excavators of the usual type is a method of raw material winning which is used in soft deposits with high natural moisture content, such as chalk or marine clay. Excavating and loading are performed in a single operation.

4 Loading

4.1 Development trend

The trend in loading machines in the last ten years has been steadily away from cable-operated face shovels and towards the increasing use of wheel loaders and hydraulic excavators. The diagram in Fig. 4 illustrates this development, which is representative of about 65% of German cement production.

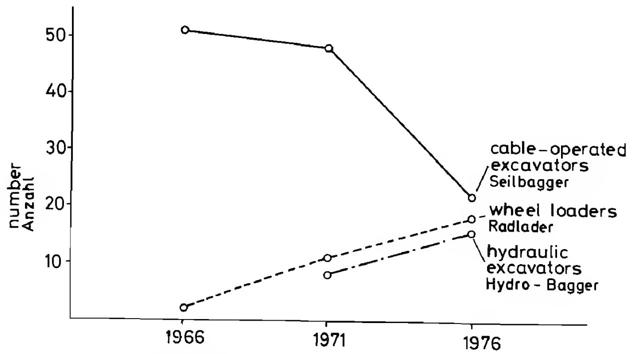


Fig. 4: Trends in loading machines

4.2 Loading machines

The machines used for loading in open-pit quarrying in solid rock, including limestone, marl and shale, are cable-operated excavators, hydraulic excavators, wheel loaders and (in special cases) crawler loaders. The choice of machine to be used in any given instance must be made with great care, because once a particular system has been adopted, a subsequent change to a different system involves heavy expenditure which may overtax the resources of relatively small undertakings. Large ones usually operate with several systems of loading machinery, enabling these to be interchanged to suit varying conditions of service.

4.2.1 Cable-operated excavators

Mechanization of loading in quarries started with the introduction of the cable-operated excavator, more particularly the face shovel, which is still on the market and available from many manufacturers and in many sizes. With its bucket fixed immovably to the arm, the diesel or electrically powered face shovel is purely a loading machine. Its relatively high capital cost can be justified by long service life, often twenty years or more. Larger machines generally last longer than smaller ones.

The electric excavator has the advantages of requiring fewer repairs and offering higher operational readiness than the diesel excavator. The latter is, however, more suitable under conditions where it has to travel fairly frequently from one working position to another. Diesel excavators are manufactured chiefly in the smaller size range. From about 2.5 m^3 bucket capacity upwards, electrically powered excavators are usually employed in the Central European countries.
Depending on its size, the electric excavator is equipped with one or more motors. In the latter case, there is often a separate motor for crowding, slewing, lifting and travelling. On large machines, loss-free control and favourable starting conditions are provided by Ward-Leonard or thyristor systems. Otherwise rheostatic control is the usual method.
The declining use of cable-operated excavators as loading machines in quarrying is attributable to several drawbacks:

- the rigidly fixed bucket requires a well fragmented rock pile suitable for loading;
- the excavator itself has poor mobility, i.e., it cannot be moved quickly and conveniently to fresh working positions (and is therefore unsuitable for selective loading);
- it is rather unsuitable for dislodging rock from a quarry face or clearing away any masses rock that have remained standing at the toe of the face.

4.2.2 Hydraulic excavators

Although hydraulic excavators have long been used in quarrying, they initially made little headway because of their small size (0.3 – 0.7 m^3 bucket capacities) and the rigid attachment of the bucket to its arm. It was only with the introduction of the movable loading bucket in lieu of the fixed bucket (actuated by hydraulic rams for tilting movements) that the advantages of these machines began to be widely recognized. The bucket of the hydraulic excavator has three degrees of freedom:

(1) raising the bucket;
(2) crowding (forward motion of the bucket);
(3) swivelling of the bucket in relation to the arm.

Hydraulic excavators mostly have a service weight in the range between 50 and 90 t, with bucket capacities of 3 to 4 m^3. Larger machines are seldom used in cement raw materials quarrying. In the open-pit mining of other minerals, however, there is a trend towards the use of machines weighing more than 100 t, with buckets of 6 to 8 m^3. The buckets may be of the tipping or the bottom-opening type, the latter being better suited for careful loading of the haulage vehicles, but has the disadvantage of heavier wear and the need for additional hydraulic equipment to operate the opening mechanism.
The three degrees of freedom enable the hydraulic excavator bucket to perform a swivelling movement up or down, so as to adjust the position of its teeth to obtain the best possible penetration for digging, without causing collapse of a heaped-up rock pile. Also, larger pieces of rock can be selectively scooped up from the pile. For

digging from a rock face the angle of the teeth can be suited to the direction of the strata. The excavator can in fact be used for the direct breaking-out of material from a quarry face, though of course the loading cycle time will then be increased and the performance of the machine in terms of loading rate (tonnes/hour) correspondingly reduced. However, as an adjunct to blasting, the hydraulic excavator can suitably be used for clearing and trimming the quarry floor and for removing any toe rock masses that have been left standing.
Besides this good bucket manoeuvrability and the resulting optimum utilization of the biting or break-out force, other advantages of the hydraulic excavator are its lower weight and greater mobility as compared with the cable-operated excavator. This mobility relates more particularly to its travelling capacity and also to the speed with which it performs its various operating motions. In addition, various types of bucket as well as other attachments can interchangeably be fitted to the excavator, so that it is indeed a universal machine. For example, a hydraulic hammer for secondary fragmentation of boulders can be attached, or a spade which can be operated with remote control of the excavator, so that it can be used for the trimming of quarry faces with no risk of personal injury.
On the other hand, hydraulic excavators are usually at a disadvantage in having a shorter service life and a lower degree of operational availability than the cable-operated excavator. Although the drive and hydraulic units are generally so designed as to be readily exchangeable and renewable, repairs nevertheless require more skill and care.
Hydraulic excavators are available as diesel or as electrically powered machines. The high cost of diesel fuel is a strong argument in favour of electric drive, which has the additional advantage of a higher service life expectation. On the other hand, it receives its power supply through a cable which not only limits its range of action but may also impede the movements of the haulage vehicles.
The use of hydrostatic drive in combination with power-summation control achieves favourable operating efficiency. With this method of control the power and the working speed can be adapted to the working conditions, while the oil pressure in the dual circuit hydraulic system plays a major part in applying the appropriate force in performing the required motion (bucket, slewing gear, bucket arm, boom, travel machinery). The rate of oil supply is the deciding factor for the speed with which the motion is performed.

4.2.3 Wheel loaders

The wheel loader, or wheel-mounted loading shovel, has been further improved in recent years. In respect of mobility it is far superior to the excavator and is more particularly suitable for selective quarrying where the loader has to serve several loading points, sometimes rather widely separated, all within short intervals of time. Besides carrying out rock loading duties in the quarry, the wheel loader is suitable for clearing and trimming work as well as for other handling and loading duties in the cement works itself.
Most of these machines used in the cement industry have bucket capacities of between 3 and 8 m^3. About 80% of all these machines employed in rock quarrying,

and 100% of those with more than 2 m^3 bucket capacity, have articulated frames and are equipped with centre pivot steering. Such machines are more manoeuvrable and attain higher loading rates than rigid-framed wheel loaders of equal bucket capacity. Because of the travel movements that the loader has to perform between scooping up the material and depositing it in the haulage vehicle, its working cycle time is longer than that of the excavator (which does not change its position during the loading operations), though this drawback can be compensated by the use of larger bucket capacities. The travel movements cause heavy wear on tyres. Efforts to improve tyre service life include the use of tyre chains for protection against cuts by sharp pieces of rock. Another development with the same purpose is the so-called beadless tyre, which has a carcass formed as an oval-section air chamber, to the circumference of which a renewable fitting belt is attached. U-shaped shoes are bolted direct to anchor eyes vulcanized into the belt. Better traction grip, protection of the tyres from damage by cuts and the elimination of overheating are advantages claimed for this tyre system.
For successful use of wheel loaders the rock pile should be well fragmented, as the ripping and break-out forces that such machines can develop are only about one-sixth to one-third of those of comparable excavators. The wheel loader is thus unsuitable for the loosening of rock, a circumstance which limits its use as a loading machine in conjunction with quarrying by surface blasting, for example.
On the other hand, besides being used purely for loading fragmented rock into trucks, the wheel loader can also be used for transporting this material over limited distances – up to about 100–150 m ("load and carry" operation).
The service life of a wheel loader is shorter than that of an excavator. The mechanical and hydraulic systems of the articulated wheel loader with centre pivot steering are sophisticated and subject to severe operating loads and stresses, requiring a correspondingly large amount of servicing and maintenance. Against this the initial cost of the machine is relatively low, and when used for "load and carry" duties it enables savings in haulage vehicles and personnel to be effected.
When digging into rock pile consisting of jagged interlocking fragments, the wheel loader will have to develop its maximum digging force, which may exceed the overturning load of the machine, so that its rear wheels tend to lift off the ground. Extra counterweight to prevent instability can be obtained by filling the tyres with water.

4.2.4 Crawler loaders

In quarrying, the use of these machines is generally confined to sites where the ground is very rough or very soft, e. g., in open-pit clay digging. The bucket is mounted on a crawler undercarriage which can function under these unfavourable conditions. On the other hand, the travel movements are slower than those of a wheel loader and the cycle time (and therefore the loading rate) correspondingly less favourable. Equipped with a ripping attachment, the crawler loader can additionally perform light breaking-out duties.

5 Haulage

Haulage comprises the transport of the fragmented rock pile material from the loading point to the crushing plant. Two main systems are to be distinguished:

(1) haulage by rail-mounted vehicles;
(2) haulage by rubber-tyred vehicles and other means.

Depending on the choice of haulage system and the particle size of the material to be handled, the rock pile loaded by the loading machine is either fed to a primary crusher in the quarry, the product of which is further transported to the cement works, or the rock pile is carried in heavy dump trucks or railway wagons to a crushing plant located away from the quarry. Intermediate solutions are possible. Thus, the rock may be loaded into trucks and taken to a primary crusher in the quarry, the crushed material then being delivered by a belt conveyor system to the cement works. Other variants are likewise available, and the choice of haulage method will depend primarily on considerations of economy. In addition, local factors play a part, such as the haulage distances, the gradients on the haulage routes, the number of working points in the quarry, the bearing capacity of the ground, and the need for selective quarrying.

5.1 Rail haulage

With a few exceptions, haulage of quarried materials by rail-mounted vehicles has been superseded in recent years. "Railless" haulage, mainly by dump truck or belt conveyor, is now predominant.
Traction is provided by diesel locomotives or, on larger projects, electric locomotives. The latter have the advantage of requiring fewer repairs, but are liable to cause problems and extra expense on account of the system of overhead contact wires needed for powering them. Standardized track gauges are 600 mm, 900 mm and 1435 mm. The payload per wagon is limited by the gauge, e.g., 4 m^3 for 600 mm.
It should be noted that certain minimum radii of curvature have to be complied with in laying the tracks and that the maximum gradient a loaded train of wagons can negotiate (over short distances) is 1:17.

5.2 Haulage by rubber-tyred vehicles and other means

5.2.1 Heavy trucks

Trucks as the principal means of haulage were first used in open-pit mining in America in 1937. Those vehicles were of 15 to 20 t payload and engine power rating up to 110 kW (150 h.p.). Developments since those days have led to trucks with load capacities of up to 318 t and powered by locomotive diesel engines that can develop 2427 kW (3300 h.p.).
The vehicles can be subdivided into non-articulated and articulated vehicles with two or more axles and various systems of dumping, i.e., discharging the load. The following description is confined to forward-control two-axle rear-dump trucks, the type most extensively used in cement raw materials quarrying.

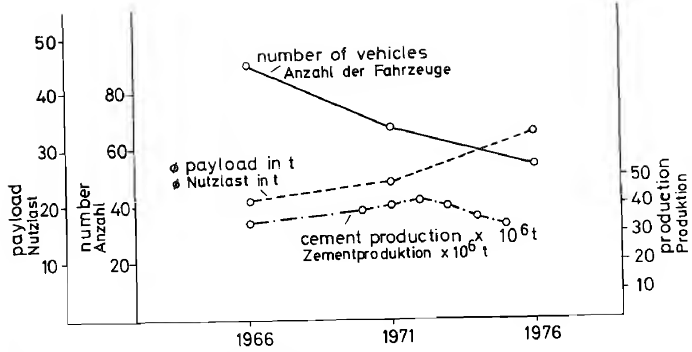


Fig. 5: Evolution in haulage vehicles

In about 90% of all open-pit rock quarries in the Federal Republic of Germany with annual outputs of over 50 000 t, in 1976 the main haulage equipment consisted of heavy and medium-duty trucks. Fig. 5 illustrates the evolution in haulage vehicle utilization by cement works representing about 65% of all West-German cement production. It appears from this diagram that the number of vehicles has steadily diminished in the last ten years, while the payload per vehicle has increased.
High-speed and low-speed diesel engines are used for powering the modern heavy trucks. The largest vehicle at present in existence, with 318 t load capacity, is equipped with a 3300 h.p. slow-running diesel. The trucks most commonly used in quarrying operations are, however, of 35 t or 50 t capacity, with engine ratings mostly between 400 and 700 h.p.
Powershift transmission is now standard equipment on medium-sized vehicles (up to about 100 t), while mechanical gearboxes are used only in the smaller types of vehicle. For heavy dump trucks (above about 100 t) mechanical transmission systems are now obsolete. These vehicles have diesel-electric drive or have direct-drive axle motors (mechanically connected to the wheels) or wheel-hub motors. The third possibility is hydrostatic power transmission.
The braking system is subject to heavy loads and has to be designed and constructed to appropriate standards of efficiency and safety. It comprises the service brake and emergency brake, an auxiliary or parking brake, and a retarder. In principle, the brakes are designed as multiple-circuit systems.
The tractive force diagram is an important basis for assessing the performance of a dump truck. It is necessary to find the optimum combination between the tractive force in the low speeds and in the highest speed ("top gear").
As the ratio of payload to unladen weight is steadily increasing and the weight of the vehicle therefore varies greatly, while the roads on which it travels are usually unpaved, the suspension has to stand up to severe conditions of service. The

requirement that springing should be equally good for the unladen and the fully laden vehicle is fulfilled by a suspension system with a parabolic spring characteristic, i.e., the curve representing the spring tension as a function of the spring travel is not a straight line but a parabola, so that the tension increases more than proportionally with increasing compression of the springs. The vehicle manufacturers strive to achieve this suspension behaviour by means of various springing and damping systems:

- hydropneumatic suspension (oil/gas);
- rubber springing systems (rubber cushions and telescopic struts or rubber-element telescopic struts);
- steel springs;
- hydraulic suspension systems.

The dump bodies, or hoppers, of the vehicles have to withstand very rough service conditions and are made of highly wear-resistant steel plate with stiffening features. At the front end there is a projecting shield to protect the driver's cab. The body can be heated with engine exhaust gas to prevent sticky materials from caking inside it under wet weather conditions. The tipping movement of the body, for dumping the load, is performed by hydraulic action.

Among the various cost items in the operation of haulage vehicles, tyre wear is especially important. The rate of wear depends on several factors, including the tread pattern and the possible use of protective chains on severely abrasive rock terrain.

The functional availability rating of a heavy truck with proper maintenance, repairs and spares planning can be put at around 80%. The condition of the haulage roads not only affects tyre wear, but also hill-climbing ability, vehicle speed and fuel consumption. As the roads are, as a rule, not surfaced with permanent paving materials, a grader is a useful machine for maintaining them in adequate condition.

With regard to the interadjustment of the loading machine and the trucks it can be said that the ratio of loading bucket capacity to truck payload capacity should be between 1:3 and 1:8 if loading is done by an excavator and between 1:3 and 1:6 if it its done by wheel loader. The outreach and loading height of the loading machines should be sufficient to ensure complete filling of the truck.

5.2.2 Belt conveyors

Encouraged by the good experience gained in lignite mining, belt conveyor systems have evolved into an important means of transport in open-pit quarrying and mining operations in loose-textured material or soft ground. In rock quarrying, on the other hand, this method of material handling is only sporadically used and then for the most part only in the production of raw materials for the European lime and cement industry.

The coarsely fragmented material produced by rock blasting has to undergo suitable primary crushing in a mobile or portable plant and has to be fed carefully onto the belt conveyor by means of a special device so as to prevent damage to the

belt. These arrangements are the main reasons why the introduction of such conveyors into quarrying is making rather slow progress. The sequence:

- drilling and blasting,
- loading,
- haulage (e.g., in dump trucks),

is replaced by:

- drilling and blasting,
- loading,
- (primary) crushing,
- conveying (belt conveyor).

Overland belt conveyor systems are usually designed for carrying the quarried materials over medium distances. These installations are characterized by flexibility of design, enabling them to adapt themselves to uneven terrain conditions, e.g., by the use of catenary-type idler sets with rollers mounted on steel wire ropes. The specific cost of transport with the belt conveyor decreases with increasing length of the system and increasing material handling rate, the latter in turn being dependent on belt width, speed, and cross-sectional (troughed) shape. The speed may be anything up to 3 m/second, and instead of a standard trough angle of 20°, more deeply troughed cross-sections with angles of 25° or 30° may be used. With increasing centre-to-centre distances the steel wire cable belt becomes the type predominantly employed. Depending on the length of the belt, its slope (angle of ascent) and handling rate, one or more drive motors, installed at one or both ends of the belt, are used to power it.

In comparison with vehicular haulage, the overland belt conveyor makes much more modest demands upon route alignment and the structures for bridging any traffic routes that have to be crossed – not least because the uniformly distributed loading of the conveyor does not require any appreciable bearing capacity of the subsoil. Gradients of up to 18° can moreover easily be overcome.

A drawback of the belt is its limited adaptability to alignments curved on plan and the susceptibility of the belt to suffer damage from coarse hard lumps of material. Furthermore, somewhat limited positional adaptability in the quarry in order to cope with varying locations of the mobile crusher (which in turn will depend on variations in the working and loading points in the quarry) is another disadvantage of the belt conveyor.

Keeping the belt conveyor in good operational order requires some monitoring devices, e.g., metal detectors and devices for the detection of tears and holes in the belt. Side guide idlers should be provided in order to assist in the training of the belt to run true and in line with the carrying idlers.

5.2.3 Load and carry

For relatively small distances between the rock pile loading point and the mobile crusher (not more than about 200 m) it may be advantageous to make use of the good mobility of the wheel loader and its favourable ratio of bucket capacity to

service weight. The currently available machines with up to 20 m^3 bucket capacity are adequate for the purpose.
In the load and carry method the wheel loader scoops up its bucket-load of fragmented rock at the quarry face and directly transports it to the crusher, which is equipped with a special receiving hopper to accept the material discharged from the bucket. With some types of crusher the loader travels up onto a kind of ramp and deposits the load into the crusher opening. The crushed product is conveyed to the cement works by overland belt conveyor.
Time studies in a limestone quarry where two wheel loaders, each of 10.6 m^3 bucket capacity, were used on load and carry duties over a distance of 100 m showed the average loading cycle time, inclusive of safety margins, to be about 120 seconds. Theoretical handling rates of up to 500 t/hour were attained, not allowing for time spent on repairs and on waiting for removal of the crusher to fresh working locations. It was found that the performance is substantially dependent on the travel speed of the wheel loader, the condition of the terrain and the gradients to be overcome. The rolling resistances encountered by the loader directly and considerably affect the performance (rate of material handling) by their reduction of the travel speed. In practice, speeds are between 6 and 12 km/hour for the laden journey and between 8 and 14 km/gour for the unladen return journey to the loading point. For a travel distance of 30 m these differences in speed between the two limits of the range on each journey may cause handling rates attained by a particular machine to vary by 29%. For a distance of 150 m the rates may, for the same reason, vary by 63%. On soft subsoil and/or on terrain with steep gradients the maximum travel distance between loading point and crusher should therefore be limited to not more than 60–80 m.

5.3 Aerial ropeways

The advantages offered by an aerial ropeway (aerial tramway) are due to its ability to overcome difficult terrain conditions. This method of transporting materials is largely independent of the nature and utilization of the ground over which the system is routed. It provides a short connection between the terminal stations and can overcome considerable gradients. Operation of the ropeway can be fully automated, while power consumption is relatively low.
Double-cable and single-cable systems are available. In the latter, one and the same cable (wire rope) serves to support as well as to tow the buckets. Ropeways can be used for virtually any distance from, say, 1 km to 100 km. The speed of the buckets is about 4 m/second.
There are some major drawbacks, however, which limit the use of ropeways to exceptional cases. The handling capacity is limited to about 500 t/hour. The capacity of an existing installation can be increased, if at all, only at considerable capital expense. Also, a ropeway system is susceptible to faults and breakdowns (especially in larger installations), while operating performance is liable to be hampered by high winds.

6 Mobile crushing plants

The combination of mobile crusher and belt conveyor system has in recent years managed only in the quarries of the European lime and cement industry to secure anything like a substantial proportion of the material handling duties. However, there have lately been moves to test and introduce this system also in other open-pit rock quarrying and mining operations. The overall trend is towards higher throughput rates. In contrast with the static crushing plant installed at the edge of the quarry (which is still the more usual arrangement), necessitating haulage of the material from the quarry face to the crusher over a distance which increases as quarrying advances, a mobile plant can be moved close to the loading point or to varying central positions in the quarry which are most favourably located at any given stage of the operations. Depending on the depth of the quarry and the length of the haulage roads, the mobile crusher in combination with a belt conveyor system may prove a substantially more economical alternative to the static crusher.
The mobile crusher is fed either directly or indirectly. With direct feed the loading machine takes up the material from the rock pile at the face and deposits it straight into the feed opening of the crusher. The best performance (highest feed rate) is obtained if the feed opening is low so that the loading machine can most conveniently discharge the contents of its bucket or hopper into it. This means that the height of the crusher and undercarriage should be suitably low.
In the indirect method the loading machine first deposits its load into a feed device which in turn discharges it into the crusher feed opening. The device should deliver the material uniformly to the crusher and may be an apron conveyor, a rubber belt conveyor or a chain conveyor. The direct feed method is used only in about 4% of all mobile crushers, the indirect method being standard practice in 96% (of which about 80% of such installations have apron conveyors, 14% belt conveyors and 2% chain conveyors).
In order to increase the throughput, the material may be screened between the feed device and the crusher, so that only the larger pieces of rock are fed to the latter, while the undersize pieces are delivered directly onto the belt conveyor.
The actual crusher may be any of the usual types of primary crushing machines. The machines manufactured in the Federal Republic of Germany for the international cement industry comprise the following types:

– single-rotor and double-rotor hammer crushers	60%
– impact crushers	30%
– jaw crushers and gyratory crushers	10%.

These figures comprise mobile as well as static crusher plants.
A swivelling conveyor may be used to receive the crushed product and provide an adaptable connecting link between the mobile crusher and the overland belt conveyor system. This intermediate conveyor is usually a belt conveyor (in 74% of the cases), or else an apron conveyor (24%) or a chain conveyor (2%).
The travel mechanism of the mobile crusher is of major importance. There are various types:

- walking mechanism;
- crawler tracks;
- rubber-tyred wheeled chassis;
- semi-mobile crusher.

The choice of the appropriate type will be based on numerous criteria, such as service weight, bearing pressure exerted on the ground, headroom (overall height), drive power rating, travel speed, manoeuvrability in different directions, performance on gradients, permissible slope on which the plant can be installed, maintenance and repair possibilities during plant operation, behaviour with regard to frequent changes of location. Fig. 6 shows the proportions of the various types of travel mechanism for mobile crushers as they have been introduced and developed over the years.
The advantages and disadvantages of these types are bound up with the conditions of use.

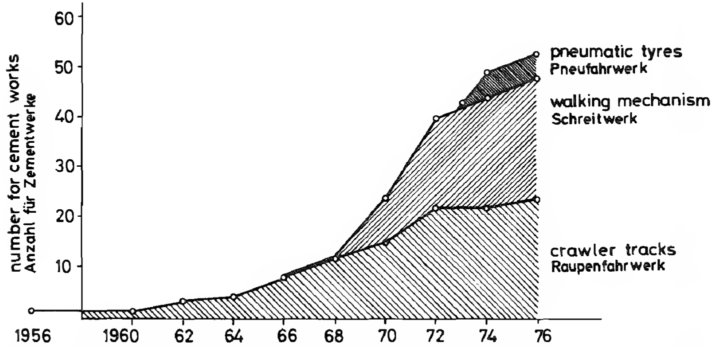


Fig. 6: Evolution in mobile crushers (Manufacturers in Fed. Rep. of Germany)

The walking mechanism is powered by a hydraulic system. Vertical rams lift the machine and its walking pad or shoe, while horizontal rams move the shoe forward and thrust the whole machine in the desired direction. This is the general principle, but actual details of the mechanism vary from one manufacturer to another.
The advantages of the walking method of travel are the low bearing pressure per unit area of ground on which the machine travels, the mobility in different directions, the ability to climb gradients, and the possibility of installing the crushing plant on sloping ground. On the other hand, this travel method is not very suitable in cases where the crusher has to be moved fairly frequently from one location to another. The travel speed is low, but so is the drive power required.

Crawler tracks may be fitted parallel or transversely to the direction of passage of the material through the crushing plant. This method of travel is advantageous in cases where the bearing pressure on the ground has to remain within fairly low limits and where frequent changes of location can suitably be achieved with moderate travel speeds. A disadvantage is the high service weight in comparison with the walking crusher, generally poorer climbing capacity on gradients, limited scope for installing the crusher on sloping ground, and inadequate mobility in different directions.
Rubber-tyred mobile crushers have advantages in terms of service weight, travel speed, and possibility of carrying out servicing while the crusher is in operation. Also, the climbing capacity, mobility in various directions, and scope for installation on sloping ground are adequate, but the high bearing pressure exerted by the wheels is a drawback.
A semi-mobile crusher has no permanently attached travel mechanism or chassis of its own. When in service, the plant is supported on a steel frame or on skids. For moving it to a different working location, a special lifting truck or a travelling chassis is used, the advantage being that these travel devices are available for use also with other semi-mobile crushers. This arrangement helps to keep down the capital cost of the crushing plant. Lifting trucks of up to 600 t capacity are now available for the purpose. Travel speeds are in the region of 2 km/hour.

7 Site restoration

In addition to the problems of environmental protection to be overcome, the pit and quarry industry has the special problem of site restoration, reinstatement or recultivation. These terms indicate the need for the raw materials quarrying or mining activities to come to terms with the demands of nature and landscape conservation. Restoration in this sense means restoring the landscape to something like its original or at least an environmentally acceptable appearance after the quarrying operations have ceased on the site concerned. Recultivation more particularly refers to creating a biologically and ecologically intact and viable natural habitat for animal and plant life.

7.1 The situation in the cement industry

The building materials industry, including the cement industry, uses raw materials which, generally speaking, are extracted rather close to the surface of the ground. These materials are found in relatively limited quantities in particular localities and can be economically transported only over fairly short distances. The choice of location for the processing plant (cement works) is therefore directly bound up with the location of the quarrying area.
The raw material needs of the German cement industry involve the quarrying of about 1 km^2 of fresh land per year. Since the Federal Republic of Germany is a country with limited raw material resources, but is one the world's largest raw material consumers, the indigenous supplies obviously must be utilized in the most efficient possible way.

7.2 Quarries and landscaping

With increasing size of individual quarries, the problems associated with site restoration have correspondingly increased.
For the present-day large and deep quarries methodically conducted restoration measures are necessary and a statutory requirement. Since it is, generally speaking, not possible to fill in the excavations because there is not enough backfill material, it is necessary to remodel the landscape in an acceptable manner. Additional changes in the appearance of the restored site will be caused by the presence of overburden tips and settling ponds.
Experience has shown that early planning for the subsequent utilization of the quarry site and ancillary features (waste tips, etc.) is essential to speedy and satisfactory reinstatement of a functional landscape configuration. What usually cannot be avoided is that the restored site will comprise exposed rock faces. The important thing, however, is the overall resulting appearance of the landscape. With methodical restoration, a varied landscape with a good range of plant and animal species can be obtained. Not only is it thus possible to restore a pleasing appearance to the countryside, but in some exceptional cases the restored site may even look better than it originally did before quarrying started. For reasons of cost, the quarry operators will strive to restore the site as soon as possible after the quarrying operations in a particular area have ended, so that topsoil spreading and recultivation can be interlinked as closely as possible.
An alternative useful quarry site restoration method is to utilize the excavations for refuse disposal, so that they are filled in before final landscaping.

7.3 Restoration features

Planning the site restoration measures involves hillsides, benches, final quarry floor, tips and settling ponds. In addition, the effect of trees and shrubs planted in connection with these measures upon the propagation of noise, waste gases and noise should be taken into consideration.

7.3.1 Hillsides

In the present context these comprise the areas situated between the rim of the quarry and the unaffected surrounding land. The plants, shrubs, etc. planted on these strips of land should protect them from soil erosion and should moreover scatter their seeds onto the benches, floors and quarry faces. Hence the hillside vegetation forms the basis and starting point for the natural flora and the associated landscaping within the quarry Any waste tips (overburden piles, etc.) that may exist on or near the hillside strips can suitably be included in the planting program. While quarrying operations are still in progress, such grassed and planted tips provide additional protection against dust and noise nuisance. The areas in question should be planted with undemanding deep-rooted species, such as sallow (low-growing willows). These not only form and hold the topsoil, but in conjunction with the successive vegetation stages of grass, plants and bushes they provide the natural habitat for subsequent other species.

7.3.2 Berms and quarry faces

After extraction of the workable mineral, berms or benches remain on the final slopes, and the correct choice of width for these horizontal ledges is important in connection with the subsequent growth of vegetation on them. As a rule, they should be 3 to 6 m wide, depending on the height above the quarry floor and unless statutory regulations require other dimensions. Against the need for suitably wide berms must be set the requirement that the least possible quantity of workable mineral should be left behind in the quarry. A compromise will therefore have to be effected.
Marl and loam may be used for filling and banking against quarry faces, because topsoil is generally not available in sufficient quantities. Soil-forming and deep-rooting plants should preferably be used, which can protect the subsoil against erosion by water flowing down the quarry faces.
Steep rock walls are unsuitable for planting with vegetation, except where sinks (dolines) filled with loamy material already exist. A certain amount of plant life will, however, gain a foothold in loam-filled crevices and at the junctions between strata and will in course of time spread to give a natural covering of greenery to parts of the rock. In any case the walls should be stable and properly trimmed. It has been found that the stability can be considerably improved by leaving a relatively thin layer of the workable deposit in situ.

7.3.3 Final quarry floor

The final floor of the quarry should generally be levelled. However, if sufficient quantities of overburden are available, artificial hillocks may be formed, which help to introduce some pleasing variety into the overall visual impression created by the restored site. Any depressions caused by overburden stripping operations should be filled in. If the final floor is dry and topsoil is in short supply, it may be necessary to provide artificial irrigation. Otherwise a certain amount of replanting will have to be carried out to make good the losses of vegetation that occur in periods of dry weather.
On the other hand, if the final floor is below ground water level, flooding may occur when the quarry pumps are stopped. A lake will then be formed, which can be a pleasing feature of the landscape in combination with the plants, bushes and trees growing on the berms and hillside strips, besides creating an environment for aquatic birds. Also, the water in the quarry can serve as a reservoir.

7.3.4 Waste tips

The recultivation of the tips (overburden and waste material piles) is normally carried out before restoration work starts on the quarry itself. Economic, technical and landscaping criteria are applicable to the operations of locating, building up and shaping the tips.
In order to keep transport costs down, waste tips are generally located as close to the quarry as possible so as to have minimum overburden haulage distances. When

the quarry has reached its final extent, it may be advantageous to dispose the tips around the quarry site, where they can serve a useful purpose in visually screening the workings and acting as a barrier curbing the emission of noise, dust and exhaust gases.
Grassing and planting the tips should begin, in the proper season, as soon as possible after they have been completed. Besides grass, other species of plant should be sown, e. g., clover, lupins, etc. After this vegetation has had time to develop, afforestation should commence with fast-growing species such as alders. The ultimate aim should be to achieve mixed planting.

7.3.5 Settling ponds

Like the waste tips, the settling ponds must also be incorporated into the restored landscape. The choice of location for these features can be an important factor in this connection. Natural depressions in the ground, hollows or old quarry workings can suitably be used for the purpose.
The outer face of an impounding dam should be planted with trees. Deep roots help to stabilize the soil. The silted-up settling pond areas should likewise be planted with trees or otherwise be used as pasture or arable land. Other possible uses are as sports fields or recreational facilities, as such areas are usually very flat.
When substantial pond areas have thus filled up, planting on them should commence as soon as they are sufficiently firm and trafficable.

7.4 Noise and dust emission

(See also Chapter H: Environmental protection)

The planting of shrubs and trees for the purpose of noise and dust emission control should be planned and carried out before the quarry is opened up. The execution of such measures may, however, run into difficulties, more particularly in open-pit projects extending over very large areas of land. Planting should in any case immediately be started along the boundaries where the final extension of the quarry workings has been reached. This will be conducive to speedy restoration of the site and its re-integration into the surrounding landscape.
Although the sound-attenuating effect of a belt of trees and bushes is often overrated, dense forest with well developed undergrowth can reduce the sound level by between 0.5 and 2.0 dB(A) per 10 m of sound transmission path through it. Obviously, it is advantageous to make the strip of forest bordering the quarry as wide as possible.
As a barrier to atmospheric pollution, especially by dust, strategically planted trees and bushes can be of real value. From this point of view it is more effective to have a belt of high trees in stepped formation or rows of trees in a staggered arrangement, in either case allowing the wind to blow through them. This form of protection is more effective than dense forest presenting a relatively impenetrable obstacle to the flow of air.

Planted areas of this kind do not themselves produce dust, and much of the dust carried into them by wind is trapped. More particularly, the slowing down of the air currents by the foliage cause them to discharge much of their dust burden. This result is more effectively achieved if the dust-laden air can penetrate into the belt of trees, so that dust precipitation takes place in or just behind it. Eddy formation in front of dense forest also results in a certain smount of precipitation, but a lot of the dust remains airborne and is carried along in the wind that sweeps over the top of such forest instead of penetrating into it.
Roughly speaking, forest with 40% penetrability achieves the best dust precipitating effect. In winter, when the trees and bushes have shed their leaves, their effectiveness is reduced to about 60% of that attained in summer.
With regard to the effect of planted areas (more particularly: belts of forest) on the distribution and objectionable action of waste gases there are four aspects to be distinguished:

- reduction of wind velocity;
- increase of turbulence;
- true filtering action by the foliage;
- physiologically beneficial effect of wind screening by the trees and bushes.

If a belt of forest is to be at all effective in the attenuation of pollution by smoke, fumes and waste gases, two conditions have to be satisfied:

- the belt of forest should rise well above the initial level at which the smoke plume spreads out;
- the distance from the trees to the source of smoke emission must not be too great.

7.5 Cost

Because of the many and varied possibilities for the subsequent utilization of worked-out quarries and their ancillary installations, and also because of the other variable factors involved (wages, etc.), no generally-valid information on the cost of site restoration can be given. For the most commonly encountered case where restoration consists of landscaping the site by the planting of trees and shrubs, however, the following figures (for German conditions in 1979) can provide some approximate guidance:

soil stripping	at least	1.00 DM/m²
supplying fill material	at least	2.00 DM/m²
spreading 0.30 m topsoil		3.60 DM/m²
spreading 0.35 m organic soil		3.15 DM/m²
planting of seedlings, incl. subsequent care		2.25 DM/m²
planting of saplings and shrubs		3.00 DM/m²
spray seeding of rubble slopes (depending on angle of slope)	2.00 to	3.80 DM/m²
quarry floor afforestation, individual trees		2.00 DM/m²
The cost per hectare may thus be of the order of		100 000 DM.

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III. Raw materials storage, blending beds, sampling stations

By D. Schmidt

1 Introduction

The intermediate storage of raw materials between the quarry and the raw mill has traditionally formed the stockpile from which a steady supply of materials for processing in the cement works has been maintained. In addition, it has in recent years become increasingly important for the purpose of pre-blending or pre-homogenizing of the crushed stone. In a few cases, final homogenization is even achieved in this way. The principle of "bed-blending" by longitudinal stockpiling and transverse reclaiming of bulk materials has already long been practised in the coal and ore mining industries, such stockpiles being known as blending beds. It is being increasingly used in the cement industry for the homogenization of raw stone or the blending of different raw materials, but also for the homogenization of clinker, blastfurnace slag and coal.
There are a number of reasons for providing intermediate storage of raw materials in the form of a stockpile:

- processing in the works is thus made largely independent of the operations in the quarry;
- multi-shift working in the quarry is rendered unnecessary by the use of high-capacity loading, haulage and primary crushing machinery;
- noise and dust emission are reduced in that they are limited to shorter periods of time;
- the stockpile safeguards the uninterrupted supply of material to feed the present-day large kilns;
- the stockpile can deal more efficiently, in terms of material handling, with sticky materials than storage in silos can;
- the stockpiling and reclaiming operations can be satisfactorily automated;
- round-the-clock operation of the finish-blending and preparation plants fed from the stockpile enables full advantage to be taken of cheaper electricity at nights and week-ends.

The following considerations are additionally applicable to a blending bed for raw material homogenization:

- better utilization of inhomogeneous raw material deposits;
- pre-blending of different raw material components is possible;
- better uniformity of the raw meal and therefore of the clinker is achieved, so that the quality of the cement is more nearly constant.

As a rule, new cement works are equipped with blending beds of various types, with or without sampling stations. Similar arrangements are provided under most modernization schemes for existing works. The following types of blending bed are to be distinguished, all of which can be designed as longitudinal (straight) or circular beds:

- Storage stockpiles

No special requirements as to pre-homogenizing efficiency are applied, and no sampling station is needed. Stacking and reclaiming the material are done by methods not involving the use of expensive and sophisticated machines.

Double stockpiles, for raw materials containing a high and a low percentage of lime, respectively, are basically similar to the single-component stockpile. The "high" and the "low" material are simultaneously reclaimed from the blending stockpiles and are used for approximately proportioning the raw mix. Further corrective materials are added ahead of the raw mills.

– Blending beds with specified target values
  Single-component blending bed: This type of blending bed is intended more particularly for the stockpiling of limestone confirming to specified characteristic values (lime standard, $CaCO_3$, CaO). The stacking operations for building up the stockpile are monitored by a sampling station. In order to ensure a good homogenizing or blending effect, the stacking and reclaiming equipment is more elaborate than that used in the ordinary storage stockpile.

  Proportioning stockpile: In this variant the required mix proportioning is obtained by the simultaneous or successive stacking of different raw material components in the same pile. The input of materials has to be monitored by a sampling station. Here, too, elaborate stacking and reclaiming equipment is essential to achieving the blending effect. In practice, however, this type of blending bed has not come into widespread use.

In general, it can be said that pre-homogenization of the raw materials can very seldom enable subsequent homogenization of the raw meal to be dispensed with. Depending on the layout of the blending bed and its equipment, some variations in the composition of the reclaimed raw materials are bound to occur, and these are passed on to the subsequent stages of processing. Such variations have to be evened out mainly by homogenization of the raw meal. In planning the installations it is therefore necessary to consider the blending bed and the raw meal homogenization system as a single whole. If the blending bed is designed to achieve a high blending or homogenizing effect, the subsequent homogenizing treatment applied to the raw meal need be correspondingly less elaborate. Conversely, if the blending bed is designed to a lower standard of homogenization, the raw meal homogenization system will have to compensate for this.

2 Bed-blending theory

2.1 Mode of operation of the blending bed

Homogenization of materials in a blending bed can be explained as follows:

The stacking (or stockpiling) system disposes the incoming raw material in the longitudinal direction of the pile by continual to-and-fro movements, so that a number of relatively thin layers of material are deposited. In this way the raw material flow is divided into quantities of $\Delta\tau$ tonnes, each corresponding to one layer. The longer-term variations in chemical composition, which depend on a particular system of working or a particular working cycle in the quarry, are thus "cut up" and superimposed one upon another in an irregular sequence.

Reclaiming the material from the pile is done transversely to the direction of stacking by what is in principle a slicing action, more particularly if a side-acting scraper is used. With this type of reclaimer the material is removed in a certain thickness all the way from the ridge to the toe of the stockpile. With a front-acting reclaimer the entire cross-section of the pile is simultaneously acted upon by the raking-down device, so that the material removed in this way cannot really be regarded as a "slice". All the same, for the present purpose, such layers of reclaimed material will likewise be conceived as thin slices.

On these assumptions, the reclaiming operations can be described as follows:

Because of the superposition of the input variations in the composition of the material stacked on the stockpile, the material reclaimed in slices at right angles to the stacking layers will be subject to certain output variations, which are of two kinds:

(a) variations within an individual slice of material (short-term deviations);

(b) variations in the average values of the slices (longer-term deviations).

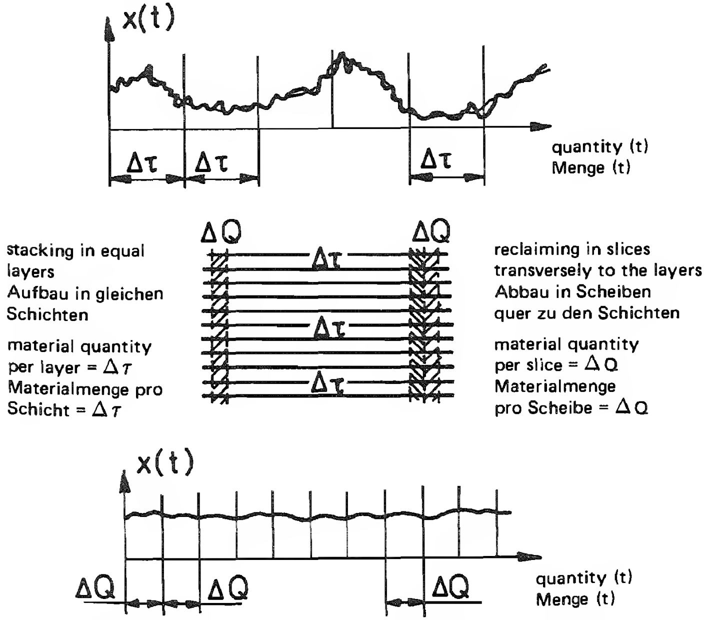

Fig. 1: Variations in the raw material composition homogenized in the blending bed

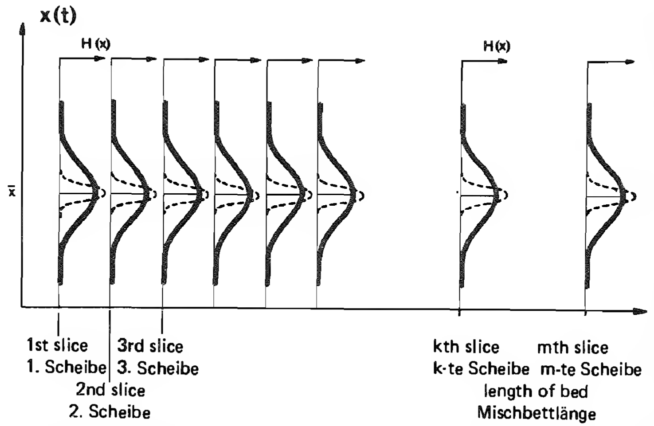

Fig. 2: Frequency distribution within the individual reclaimed slices of an ideal blending bed

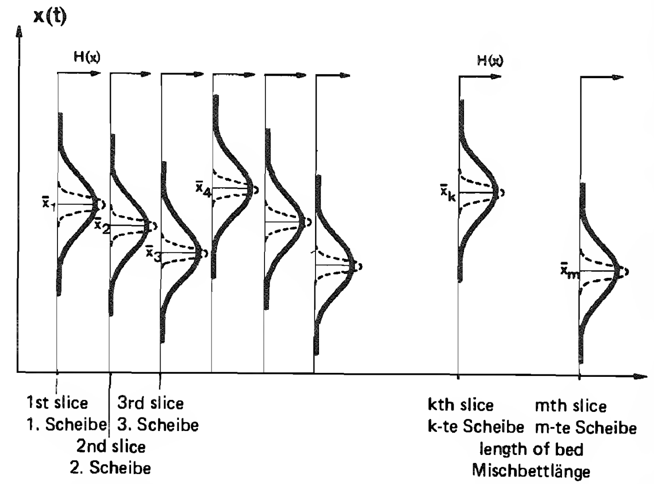

Fig. 3: Real frequency distribution within the individual reclaimed slices from a blending bed

Because of the slice-by-slice reclaiming technique the variations within the slice are evened out to a greater or less extent, depending on the type of reclaiming machine. The variations in the averages of the respective slices are predetermined by the quantities $\Delta\tau$ and the number of layers N. For correct blending bed design the quantities of material per layer and the number of layers should be so chosen that the remaining variations from one slice to another are reduced to a minimum. The blending effect can be improved – and indeed theoretically be made infinitely good – by increasing the number of layers in building up the stockpile and by using a reclaiming system that will efficiently homogenize the material.
These considerations indicate that the cycle of operations in the quarry deserves closer attention. True, the standard deviation of the input variations cannot be altered by changing the cycle, but it is possible to improve the variations from one slice to another, the more so as the quantity of material per stacked layer exceeds the loading capacity of one or more loading machines. The procedure in the quarry, e. g., the loading and haulage from several qualitatively different rock piles, may conceivably be correlated with the stockpiling of the material in the blending bed, so that quantities of material with similar characteristics of quality or composition are stacked on top of one another in the same parts of the bed. This undesirable situation, which diminishes the blending effect achieved, can be remedied by suitably varying the operations in the quarry, so as to achieve an irregular sequence of delivery of the material to the bed.
Figs. 2 and 3 show two theoretical blending bed reclaiming models. The material slices and the statistical frequency distributions of the input and output variations are shown.
Fig. 2 relates to reclaiming from an ideal blending bed. The continuous line represents the input variations, the dotted line the output variations. "Ideal" stacking of the material signifies that the chemical composition at every cross-section of the blending bed is equal to the overall average composition:

$$\bar{x}_i = \bar{x}.$$

The remaining output variations exist only within the slices of material. There are no variations between one slice and another.
In an actual, as opposed to an ideal, blending bed there additionally occur variations from slice to slice, as Fig. 3 shows. Even though the variations within the slices remain unchanged, their average (or mean) values are now no longer equal to the overall average of the chemical composition:

$$\bar{x}_i \neq \bar{x}.$$

2.2 Assessment of a blending bed

For assessing the homogenizing performance of a blending bed, the following parameters will be considered.
Four of these are introduced as estimated values of the statistical variance:

S_α^2 overall variance of the input variations
S_β^2 overall variance of the output variations

S_Q^2 variance of the output variations within the reclaimed slice
S_x^2 variance of the averages of the slices.

Other parameters to be considered are:

N the number of layers
$\Delta\tau[t]$ the quantity of material per layer
$\Delta Q[t]$ the quantity of material per reclaimed slice.

The variations within the slices (S_Q) are short-term ones. In the reclaiming operation they are evened out to a greater or less extent, depending on the type of reclaiming equipment. This can be most readily visualized when considering the action of a side-acting scraper, a reclaimer which removes the material from the pile in successive slices. The contents of each slice comprise marked variations, which are passed on to the processing equipment further down the line from the blending bed. With front-acting reclaimers the slices are thinner, the reclaiming action comprises the entire cross-section of the pile, and the variations in each slice are substantially smaller. In general, variations in the raw material are not equalized to any appreciable extent in the raw grinding plant. Hence they will have to be removed in the raw meal homogenizing system.
If reclaiming is done by side-acting machines, the output variations, i.e., the variations in the material coming out of the blending bed, will be greater than if front-acting machines are used. Therefore, with the former method it will be necessary to provide suitably effective raw meal homogenizing facilities, whereas these can be simpler if the latter method is used. If each slice of reclaimed material is regarded as a unit, the variations within it (s_Q) can be neglected, so that then only the variations between the individual slices (s_x) remain to be considered. The latter are longer-term in character. Since the averages of these raw material slices differ from the overall average, these variations cannot be removed by raw meal homogenization, but only by means of a suitable components proportioning system upstream of the raw mill.
The effectiveness of a blending bed is expressed by the concept of "homogenizing effect" (e), namely, the ratio of the standard deviations of the raw material characteristics on entering and leaving the blending bed respectively; thus·

$$e = \frac{s_\alpha}{s_\beta}$$

However, this criterion by itself is not sufficiently informative. In addition, the absolute values of the output variations should be available. In the planning of new installations these values determine the performance requirements applicable to the proportioning devices before the mills and/or to the homogenizing equipment for the raw meal. With a well designed bed-blending system it is possible to achieve good homogenizing effects, by which is more particularly to be understood: low final variations in the chemical composition of the material, despite possible high input variations of the material stockpiled in the bed.
The homogenizing (or blending) effect of a blending bed depends on the method of stacking and on the characteristics of the reclaiming machinery. Once these two

aspects have been determined, and the appropriate thickness or number of layers to build up the bed has been decided, the homogenizing effect of the stockpile as a whole will have been predetermined. The final variations are bound up with the output variations in the material reclaimed from the bed. In other words: when the design and operation of a bed have been fixed, the homogenizing effect it achieves will be constant. Higher input variations will result in higher output variations.

2.3 Estimating the homogenizing effect in advance

The blending bed design methods reported in the literature are mainly concerned with the variations between the slices of reclaimed material. The following method is generally employed. It presupposes that the raw material values of the individual slices conform to a normal distribution and are statistically independent.
For estimating the output variations the following relation is available:

$$s_x = \frac{s_\alpha}{\sqrt{N}}$$

Theoretically, high homogenizing effects can be attained by making the number of layers large enough, i.e., using very thin layers.
On the other hand, the presupposed statistical independence diminishes with decreasing layer thickness, for the quality characteristics of adjacent raw material layers stacked in the blending bed tend to be correlated with one another. This phenomenon can most easily be visualized at the reversal points of the stacking operation. At the end of each forward pass and the beginning of each return pass of the stacker, material possessing the same properties is stacked in two successive layers. Thus the condition that the material must be stacked in discrete quantities $\Delta\tau[t]$ is fulfilled only after every second layer, i.e., only every second layer contributes to the homogenization achieved in the blending bed.
The relationship between the standard deviation and the number of layers is shown in Fig. 4 for single-component and multi-component blending beds. It emerges that a worthwhile reduction in the output standard deviation is attained only if there are at least about 50 layers. With increasing number of layers the rate of improvement diminishes, so that from about 500 layers onwards there is hardly any further improvement in the homogenizing or blending effect, while the sheer technical effort and expense of building up the bed in so large a number of layers would not be commensurate with the advantage gained.
The following conclusions are to be drawn from all this:

- Predictive estimates in accordance with the method indicated above are to be regarded only as approximate. With increasing number of layers the results found for the homogenizing or blending effect increasingly tend to overestimate the effect. (This has been verified by check calculations based on accurate measured data.)

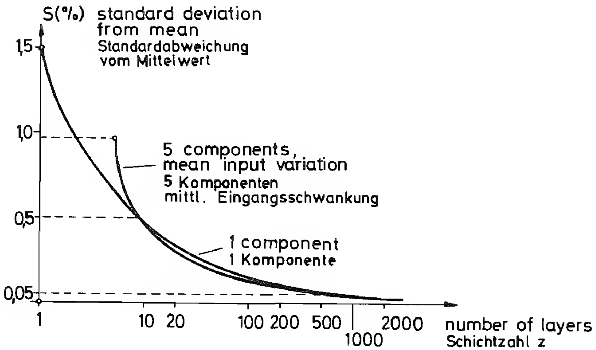

Fig. 4: Blending effect as a function of the number of stacked layers of material

- When using the formula $s_x = s_a/\sqrt{N}$ it is advisable to introduce only half the actual number of layers.
- There is no point in using fewer than 50 or more than 500 layers.

Attempts to make more accurate predictions of the homogenizing effect of a blending bed usually fail for the following reasons:

- the input variations in the material coming from the quarry are not known and are then mostly over-estimated;
- the thickness of the layers stacked in the blending bed is not constant, this being due to variations in performance of the handling and stacking systems;
- the bed-blending stockpile comprises two end cones where the conditions are different from those in the rest of the pile and which, depending in part on the particle size of the material to be homogenized, have a marked detrimental influence on the homogenizing effect.

Despite all its imperfections, the method of predicting the homogenizing effect as outlined above is now widely used.
According to Hasler, experience to date shows that with present-day bed-blending technology the following values of the homogenizing effect can be obtained:

- e = 3 to 6 if the overall variations are considered (long-term and short-term output variations);
- e = 6 to 15 if the short-term variations are left out of account (i.e., ignoring the variations within each slice).

3 Machinery and process engineering methods

3.1 Stacking methods

3.1.1 Chevron method

The raw material is deposited by a stacking device moving continually to and fro over the longitudinal centre-line of the stockpile. In this way individual layers containing equal quantities of material are disposed one upon another in the shape of a series of ridged roofs. This means that, subject to ignoring the short-term variations, all cross-sections have the same composition. The material discharged from the stacker slides and rolls down the sides of the pile, thus causing a degree of particle segregation depending on the properties of the material concerned. The coarser particles will tend to accumulate at the base of the pile. The arrow (Fig. 5) indicates that building up the pile requires only one central throw-off point of the stacking device in the longitudinal direction and can be achieved with relatively simple equipment.

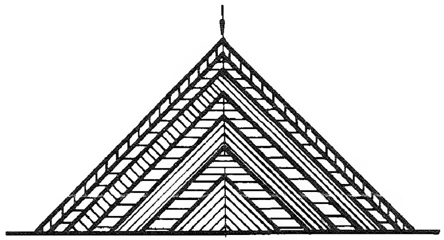

Fig. 5: Chevron stacking method

3.1.2 Windrow method

The drawback of particle segregation can be avoided by using the windrow method of stacking, in which the layers ("windrows") are disposed longitudinally over and beside one another. Although some segregation may occur during the stacking of the individual rows, it is limited to each individual row. Besides, this effect can be minimized by appropriate choice of the height and spacing of the rows of stockpiled material. The larger the number of rows, the more favourable will be the particle size distribution in the pile.

In actual practice, however, the windrow method in its pure form, as illustrated in Fig. 6, is hardly ever employed. Much more often a combination of this method and the chevron method is adopted.
A drawback of the windrow stacking technique is that it requires several throw-off positions, necessitating expensive slewing boom stackers.

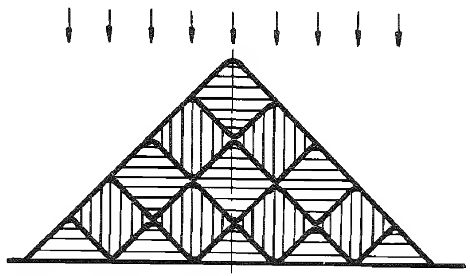

Fig. 6: Windrow stacking method

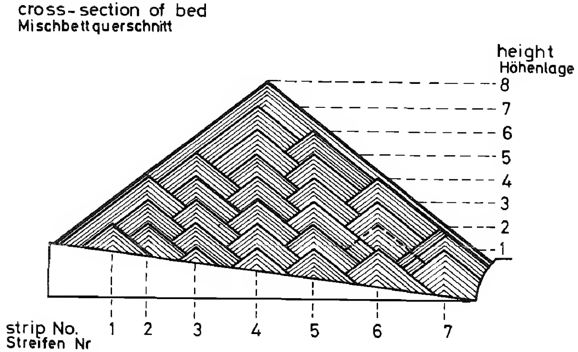

Fig. 7: Actual stacking in a blending bed

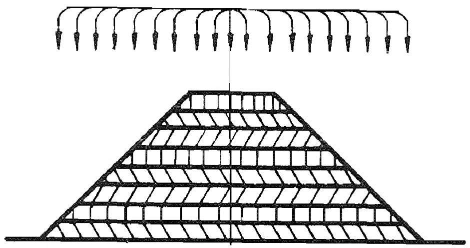

Fig. 8: Stacking in horizontal layers

3.1.3 Horizontal layers

Step-by-step advancing of a bridge stacking system in conjunction with continual slewing of the stacker belt conveyor on its boom will produce a stockpile whose individual layers are disposed horizontally one on top of another. With this method, bulk materials differing in their angle of repose and consisting of particles in relatively wide size ranges can be stacked in layers varying in thickness, without appreciable segregation.
It is also a suitable method for circular stockpiles, the stacking being done by means of a belt whose throw-off point moves in a meandering path.

3.1.4 Strata method

In terms of cross-sectional distribution of the material in the stockpile, this method is equivalent to the preceding one, but with the drawback of a certain amount of segregation due to accumulation of the coarser particles at the bottom of the pile. The layers in this system are inclined at an angle of about 32° to 38°.
This type of stockpile is especially suitable for reclaiming by side-acting machines.

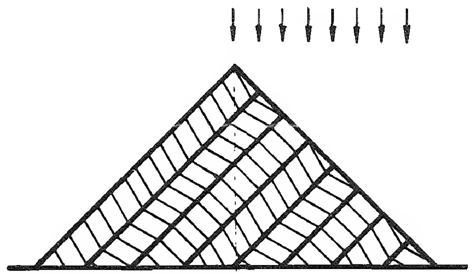

Fig. 9: Strata stacking method

3.1.5 Cone-shell method

As contrasted with the methods so far described, in which the stacking device travels continually to and fro, in the cone-shell method the stacker – a belt conveyor that can be moved along the length of the pile or a fixed-boom stacker – forms a series of conical piles heaped one against another. As soon as such a pile has been built up to the appropriate height, the stacker moves on. In this method a distinction is to be drawn between continuous stacking and alternate stacking (Figs. 10 and 11).
The homogenizing or blending effect achieved with this method is less good than that achieved with the methods described above, while there is a further

disadvantage in that the coarser particles tend to accumulate at the base of the pile. Reclaiming can be done only by side-acting scrapers or by underfloor extraction.

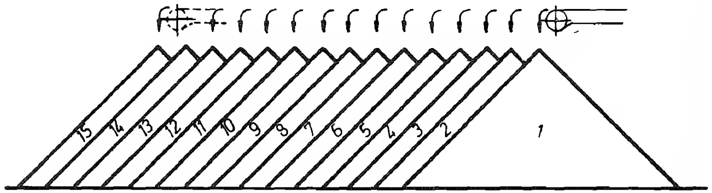

Fig. 10: **Continuous stacking method (numbers denote sequence)**

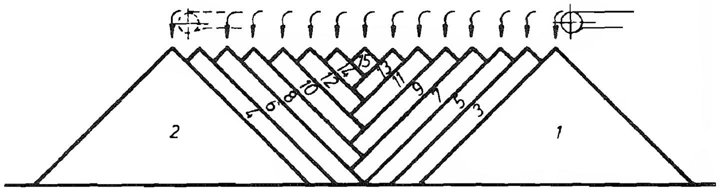

Fig. 11: Alternate stacking method (numbers denote sequence)

3.1.6 Chevcon method

The firm of PHB offers this method as a hybrid of the chevron and the cone-shell method. It is suitably only for circular stockpiles.
The stacking procedure is similar to that used in the chevron method, but instead of remaining over the ridge of the pile, the throw-off point of the stacker is varied a radial distance ΔL in the course of each to and fro cycle. The slope of the face from which reclaiming will subsequently be done can be varied by appropriate alteration of ΔL. For constant stockpiling rates the angle α of the slope will then remain unchanged (see Fig. 12).
This method very effectively overcomes the "end-cone problems" that are associated with bed-blending stockpiles and will be further discussed later in this chapter. In addition, thanks to the overlapping of old and new material in the pile, long-term variations or the effects of possible sudden changes in incoming batches of raw material can be cancelled. The number of layers comprised in each slice taken by the reclaimer is about 30% greater $(n + k)$ than the number (n) sliced in reclaiming from the chevron pile with its flanks sloped at the natural angle of repose of the material. As a result, a better homogenizing effect is obtained. Also,

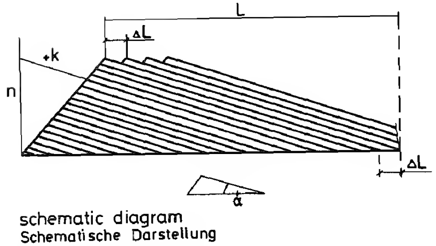

Fig. 12: **Chevcon stacking principle**

there is very little segregation into coarser and finer particles. The stacker belt can continue to deposit the incomimg material even in the immediate vicinity of the reclaimer, so that utilization of the full capacity of the blending bed can be attained at all times. Since stacking is done continuously, a circular bed built up on this principle can justifiably be called an infinite blending bed.

3.2 Stacking and reclaiming machines

Over the years, a large number of systems and machines have been developed, and from these have evolved certain types of blending bed, which will be described here.
The combination of chevron (or, where applicable, chevcon) stacking with front-acting reclaimers is to be regarded as the most favourable procedure, as it involves the least expenditiure on machinery. The process engineering disadvantages associated with the chevron method of stacking are cancelled by the use of front-acting reclaimers. Such reclaiming machines are used also for stockpiles built up in horizontal layers.
The alternative system consists in windrow stacking with reclaiming by means of side-acting scrapers. The MIAG "step-back" method is more particularly suitable for this purpose.
Good homogenizing or blending effects are also attained with the strata method and side-acting scraper reclaimers.
The homogenizing effect of bed-blending stockpiles built up by the cone-shell method and operating with side-acting scrapers or underfloor extraction is poor, and for this reason it is a system little used for blending beds.

3.2.1 Chevron stacking and end-on reclaiming

3.2.1.1 Stacking machines

Blending beds may be of the outdoor type or be accommodated in suitable buildings. In the latter case the material can be stacked by belt conveyors mounted under the ridge of the roof or by mobile floor-mounted stackers travelling the

length of the building. The arrangement and installation of belt conveyors will depend on the type of roof construction. These handling devices have the advantage of being relatively inexpensive and not taking up so much space as a floor-mounted mobile machine, so that the cross-sectional dimensions of the building can be correspondingly smaller. A disadvantage associated with belt stacking is the large height of fall of the material onto the pile. With dry material this can throw up much dust.

Stackers with fixed or movable booms (which can be raised and lowered) are used for covered as well as for outdoor blending beds. For reasons of cost it is not a good idea to install permanently mounted belt conveyer systems over outdoor stockpiles. A drawback associated with fixed-boom stackers is that dust nuisance may arise, and the attachment of telescopic discharge spouts or similar devices to

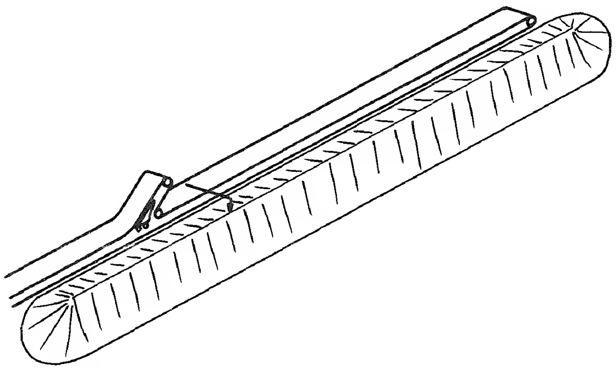

Fig. 13: Longitudinal stockpile with overhead stacker belt and tripper

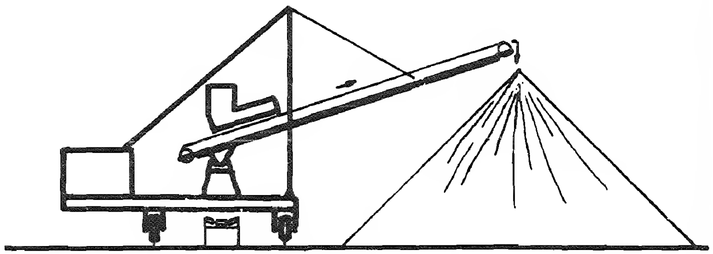

Fig. 14: Boom stacker with fixed boom (longitudinal stockpile)

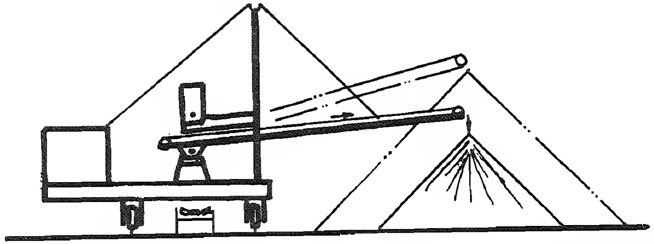

Fig. 15: Boom stacker with movable (luffing) boom (longitudinal stockpile)

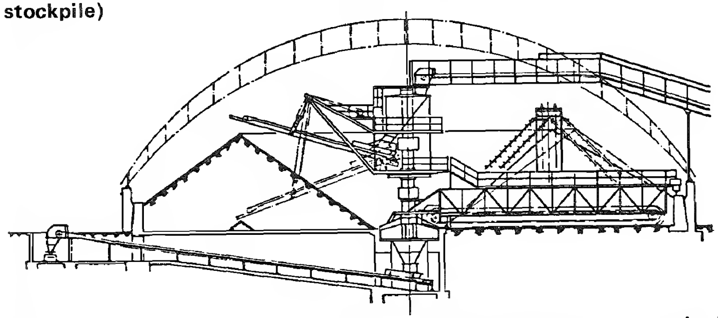

Fig. 16: Stacking in a circular blending bed with simple chevron method

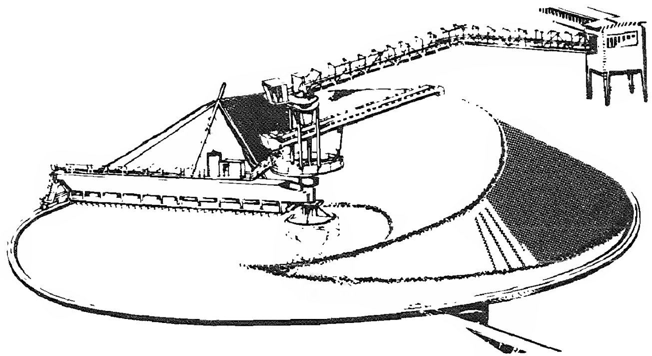

Fig. 17: Circular blending bed with "infinite" stacking on the chevcon principle

combat this nuisance is not without problems. It is preferable, under such circumstances, to employ a movable-boom stacker enabling the height of free fall of the material to be kept down to a minimum.

3.2.1.2 Reclaiming with front-acting machines

All front-acting reclaimers, i. e., machines for "end-on" removal of material from stockpiles, are equipped with some form of handling device which is only able to carry away the material from the toe of the pile. The material is dislodged from the pile by the action of a raking-down device which sweeps across the cross-sectional face. Each cycle of the device removes a thin "slice" comprising all the layers in the pile, and in the process of sliding and tumbling down the sloping face the material of the various layers is mixed together. To obtain a good homogenizing effect it is of course essential for the raking-down device to involve the entire face of the pile.

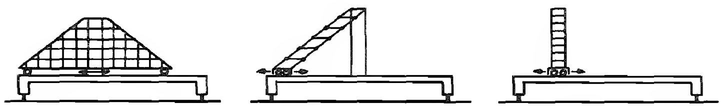

Fig. 18: Raking-down devices

These devices are of various kinds (Fig. 18):

- Harrows are triangular structures fitted with renewable teeth and so inclined as to suit the angle of repose of the stockpiled material. The latter is dislodged by the to-and-fro movement of the harrow across the face of the pile.
- The rope-operated scraper comprises two ropes which pass around pulleys at the top a frame near the apex of the pile and are attached to a slide or carriage which moves to and fro on the supporting bridge. As a result of this shuttling motion of the slide, the ropes perform movements somewhat like those of a car windscreen wiper and thus dislodge the material from the entire face of the pile. For dealing with difficult material, the two ropes may be interconnected by pivotably attached cross-members fitted with teeth, thus substantially increasing the loosening effect and reducing wear on the ropes.
- Scraper chains are more particularly appropriate for dealing with very difficult material requiring considerable effort to dislodge it from the face of the pile. In the course of its to-and-fro movement the scraper chain sweeps across the entire face and actively scrapes the material down to the toe.

Associated with these raking-down systems there are, in the main, four different types of front-acting reclaimer; these are illustrated in Figs. 19 to 22.

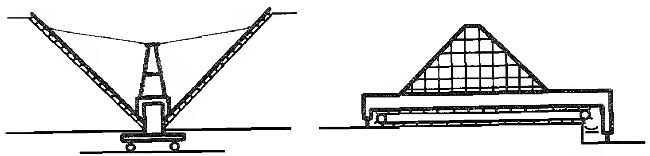

Fig. 19: Bridge-type scraping reclaimer with harrow attachment

Bridge type scraping reclaimer:

The bridge on which the raking-down device is mounted accommodates a scraper chain conveyor whose blades shift the dislodged material along to a collecting belt conveyor that extends along one edge of the stockpile.

Advantages:

- Good homogenizing action because thin slices are removed from the entire cross-sectional area of the pile.
- The rate of reclaiming and handling of the material is constant and quite simple to regulate.
- The machine takes up only a modest amount of cross-sectional space inside a building.
- The direction of reclaiming can conveniently be reversed.

Disadvantages:

- There is an upper limit to the handling rate.
- Along the edge of the pile beside the collecting conveyor a scrape feeding shelf has to be provided, which must not be covered with material during stacking and which thus restricts the utilizable stockpiling width

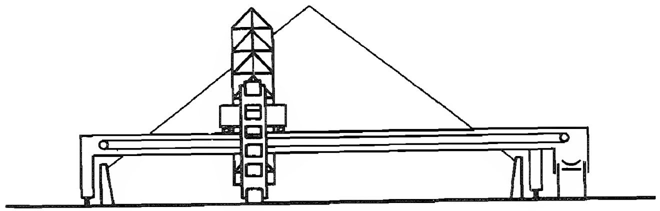

Fig. 20: Bucket-wheel reclaimer, bridge-mounted type

Bridge mounted bucket-wheel reclaimer.

This type of machine comprises one or more bucket-wheels and a raking-down device which together are moved to and fro on the bridge across the face of the stockpile. The material dislodged from the pile is scooped up at the toe of the face by the bucket-wheel

Advantages:

- Good homogenizing action. The variations due to the fact that the material is not constantly taken from the entire face of the pile during the to-and-fro cycle of the bucket-wheel are of short duration and can without difficulty be averaged out in the subsequent processing stages.
- The handling rate is virtually unlimited. For high rates, two or more bucket-wheels can be mounted on the same bridge.
- Sideways transport of the material to the longitudinal collecting belt conveyor is done by a belt installed in the bridge. This arrangement saves energy in comparison with the scraping reclaimer and moreover enables a low toe wall to be constructed as a lateral boundary to the stockpile. Thus there is no risk of overfilling the pile, while the amount of space occupied is kept down to a minimum.

Disadvantages:

- The rate of material handling during the transverse movement of the bucket-wheel is not constant. This is compensated by using three different transverse travel speeds for the bucket-wheel which are applied at successive stages of its to-and-fro cycle. The drawback is that such three-speed operation requires more elaborate control arrangements.
- The cross-sectional space requirements are greater than those of the bridge-type scraping reclaimer.
- On reversal of the reclaiming direction the buckets have to be turned over.

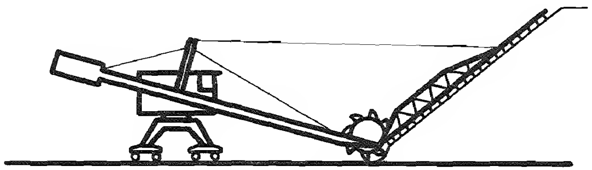

Fig. 21: Bucket-wheel reclaimer with slewing boom

Bucket-wheel reclaimer with slewing boom.

The bucket wheel is mounted at the end of the boom which swings to and fro across the stockpile, so that the reclaiming face is slightly curved. In other respects the action is similar to that of the bridge-mounted bucket-wheel reclaimer.

Advantages:

- These are as already mentioned for the bridge-mounted machine.
- The track rails are within the width of the pile (and buried by it), so that a further saving in space is obtained.

Disadvantages

- These are also as mentioned for the bridge-mounted machine.
- Turning the machine round requires much space, which can be a serious drawback inside a building.

Drum reclaimer:

The material dislodged from the face of the stockpile is picked up by scoops mounted on a revolving cylindrical drum or tube and is deposited onto a belt conveyor installed inside the drum. This type of machine is characterized by its good homogenizing or blending effect, since the entire width of the pile is at all times being acted upon. However, the elaborate and expensive design features make the drum reclaimer uneconomical except for very high rates of handling (above 2000 t/hour).

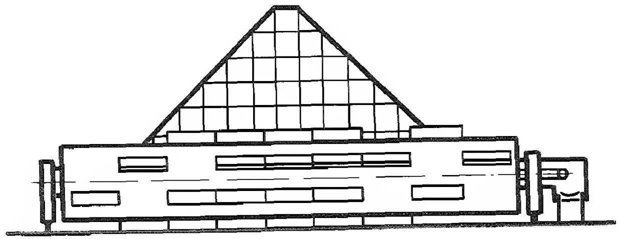

Fig. 22: Drum reclaimer

3.2.2 Blending bed system with windrow stacking

3.2.2.1 Stacking machines

Stacking by the windrow system can, like chevron stacking, also be done in buildings with ridge-mounted belt coveyors and appropriate transverse belt conveyors, but all the disadvantages already mentioned – e.g., great height of fall, etc. – are applicable in this case too. It is better to use boom stackers for building up the stockpiles. There are three types of such machines:

(a) stacker with fixed boom and telescopic belt conveyor;
(b) stacker with movable boom (luffing motion) and telescopic belt conveyor,
(c) stacker with movable boom comprising luffing (raising and lowering) and slewing motion.

Type (a) has the disadvantage that the material falls from a great height, just as it does from a belt conveyor mounted under the ridge of the roof of a building. With a boom stacker of type (b) the height of fall can be kept down to a minimum. Finally, type (c) is a universal machine, which is more particularly suitable where two

parallel stockpiles have to be formed side by side, in which case the stacker travels longitudinally between the piles, these being built up as required, by slewing the boom in either direction.

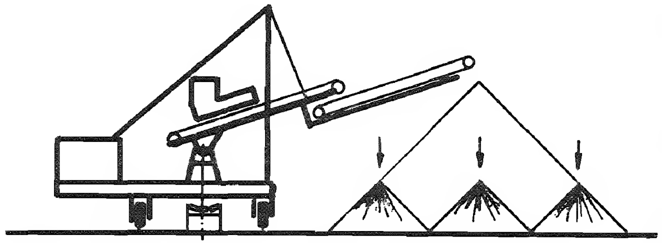

Fig. 23: Boom stacker with fixed boom and telescopic belt

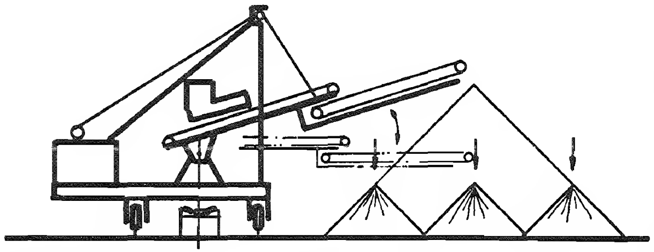

Fig. 24: Boom stacker with movable (luffing) boom and telescopic belt

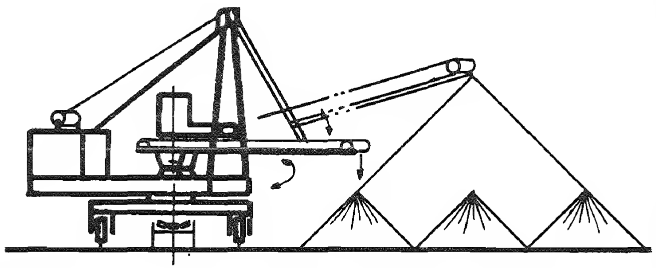

Fig. 25: Boom stacker with movable (luffing and slewing) boom

The windrow system of stacking can be applied to any type of blending bed, i. e., straight or circular. In the latter case it must be borne in mind that the outer rows contain more material than the inner. Another advantageous method of stockpiling consists in depositing the material in large homogenizing troughs or tanks such as those constructed by the engineering firms of FLS and MIAG.
The windrow method applied to a circular blending bed is illustrated in Figs. 26 and 27.

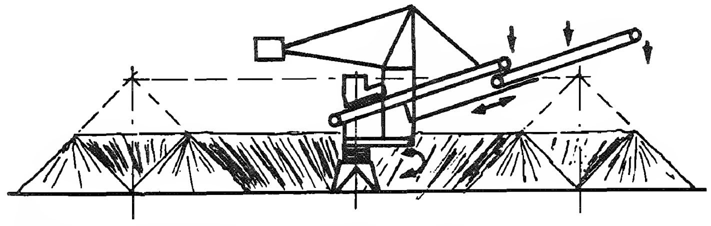

Fig. 26: Boom stacker with luffing boom and telescopic belt conveyor for a circular stockpile

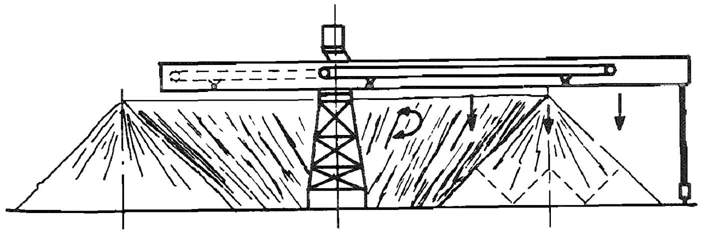

Fig. 27: Slewing bridge with movable belt conveyor, supported on central tower and external rail

3.2.2.2 Reclaiming by side-acting scrapers

Any of the front-acting machines described in 3.2.1.2 can be used for reclaiming from bed-blending stockpiles built up by the windrow method. Such machines would certainly effect some further, though slight, improvement in the homogenizing effect. However, the use of scraper chain reclaimers, operating either as front-acting or side-acting machines, has become established practice for such blending beds. More particularly the so-called step-back method of MIAG has been developed for the purpose. The material is reclaimed from one side of the pile by a machine which travels short distances to and fro in conjunction with raising and

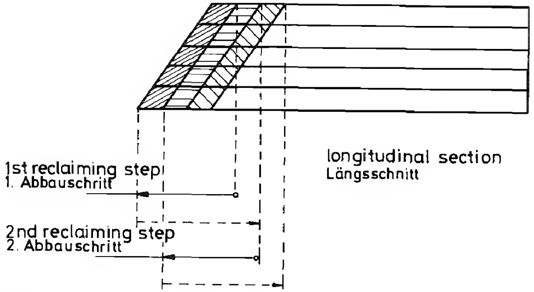

Fig. 28: MIAG step-back reclaiming principle

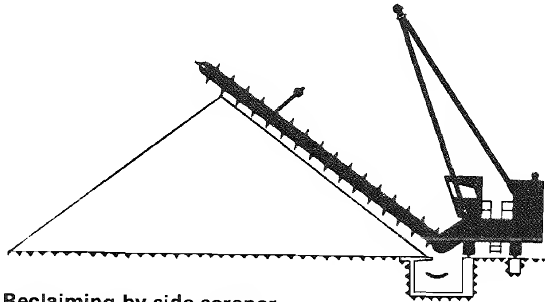

Fig. 29: Reclaiming by side scraper

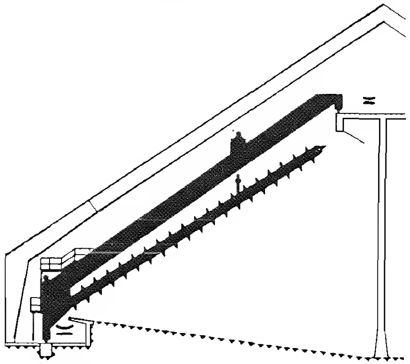

Fig. 30: Reclaiming by semi-portal scraper

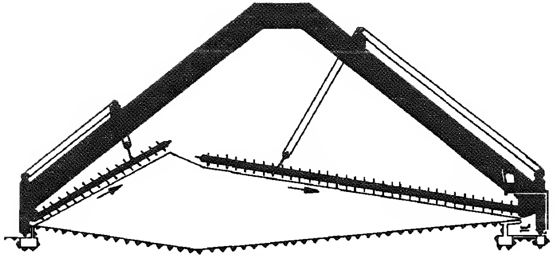

Fig. 31: Reclaiming by portal scraper

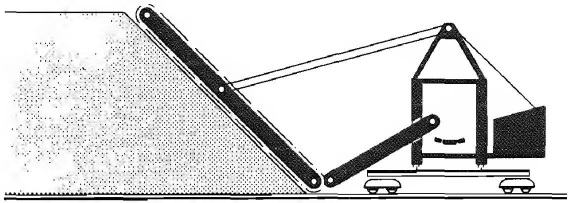

Fig. 32: Reclaiming by front end scraper

lowering of the scraper arm. During reclaiming from the top to the toe of the pile the reclaimer travels slowly back, so that the face of the pile is scraped away in the shape of a helically curved surface. The effect achieved in this way is similar to that of reclaiming with a front-acting machine. The reclaiming action does not comprise the entire face simultaneously.

Reclaiming with side scrapers has the disadvantage that the rate of flow of the reclaimed material is not constant, while the homogenizing effect is less good than that obtained with front-acting machines.

3.2.3 Blending bed systems with horizontal and inclined stacking

3.2.3.1 Stacking machines

A bed-blending stockpile can be built up in horizontal layers by means of a slewing-boom stacker or an overhead belt conveyor (mounted under the ridge of the roof) with slewing throw-off belts. A luffing-boom stacker or a ridge-mounted belt conveyor with simple transversely movable throw-off belts can alternatively be used for building up stockpiles consisting of inclined layers. These stacking machines have already been described. Stacking in horizontal layers is widely used

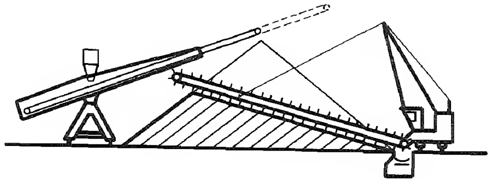

Fig. 33: Travelling scraper and strata stacking method

also for depositing materials into trough or tank type homogenizing systems. Stacking in inclined layers by the strata method, and reclaiming by a slow-moving side scraper, are illustrated in Fig. 33.

3.2.3.2 Reclaiming machines

Scraper chains are unsuitable for reclaiming from a stockpile built up in horizontal layers. Front-acting machines, as described in 3.2.1.2, must be used for the purpose. Reclaiming from piles with inclined layers may be done not only with front-acting but also with side-acting scrapers.
Reclaiming from homogenizing troughs can be done with scrapers or with bucket-ladder (or chain-bucket) machines.
The homogenizing effect obtained with a trough type blending bed is generally better than that obtained with a blending stockpile built up by the strata method. Both systems are to be rated as very efficient, however

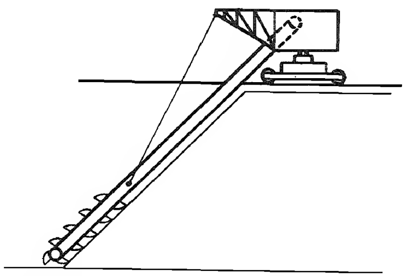

Fig. 34: Homogenizing trough with bucket-ladder reclaimer

3.2.4 Blending bed based on the cone-shell method

This method, illustrated in Fig. 35, calls for no special comment. The stacking and reclaiming machines are similar to those already described. In terms of homogenizing effect this is not a good system, however.

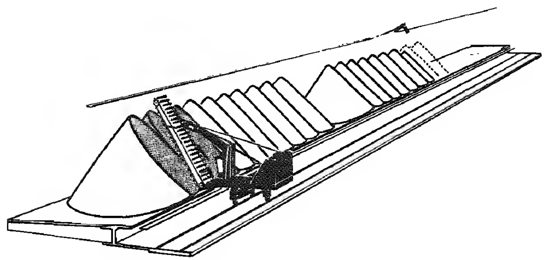

Fig. 35: Stacking and reclaiming of a blending bed based on the cone shell-method: overhead stacker belt, longitudinally travelling scraper reclaimer

3.3 Arrangement of blending beds

As already stated, a distinction is to be drawn between longitudinal (straight) stockpiles and circular stockpiles, while the trough or tank type, in which the material is stored substantially below ground level, is a third main variant. With the longitudinal arrangement the blending bed will generally comprise two stockpiles, worked discontinuously in that one is being built up by the stacking equipment while material is being reclaimed from the other. On the other hand, with a circular pile the two operations – stacking and reclaiming – can proceed simultaneously on the same pile, one end of which is being built up while the other is being reclaimed, so that these operations can proceed continuously. The chevcon method is more particularly suitable for circular blending beds, in which case virtually the entire capacity of the pile is effectively available and the stacking and reclaiming operations proceed in an "infinite" cycle.
In the case of homogenizing troughs the two operations – stacking and reclaiming – are usually carried out simultaneously in that reclaiming takes place in one part of the trough while stacking proceeds in another part of the same trough.
In deciding which layout to choose for a bed-blending system the following considerations are applicable:

- how much space is available for accommodating the bed?
- what scope for possible future extension is there?
- do subsoil conditions (bearing capacity) have to be taken into account in planning the bed?

It is not possible to make any generally-valid statements as to the size (capacity) of the stockpiles, as the blending bed for each cement works has to be laid out to suit the particular requirements of the case. Roughly speaking, however, it can be said that a stockpile should contain about one week's supply of raw material for the cement works.

3.3.1 Longitudinal stockpiles

Longitudinal (straight) bed-blending stockpiles may be arranged either parallel or in line.

3.3.1.1 Parallel stockpiles

Advantages:

- Moderate length/width ratio on plan.
- Fits in easily with the layout scheme for a cement works.
- Capacity can easily be increased.

Disadvantages:

- Reclaimer has to be changed over from one pile to another.
- Either a slewing stacker or a stacker with two booms is needed.
- Large number of belt conveyors and transfer points.
- Long roof spans for stockpiles accommodated in a building.
- Extra space required for change-over of machines.
- End-cone problems.

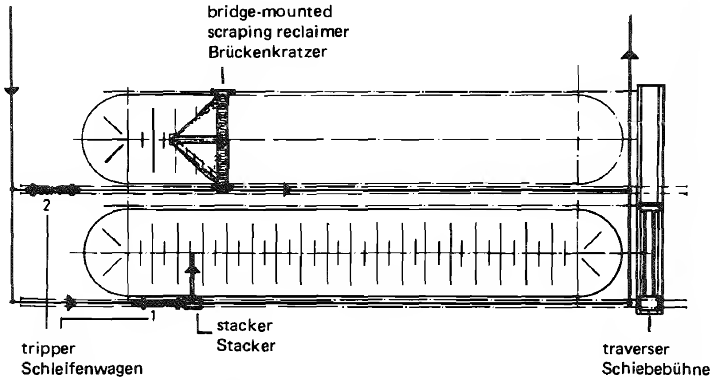

Fig. 36: Parallel stockpiles

3.3.1.2 In-line stockpiles

Advantages:

- No change-over of machines.
- No slewing stacker required.
- Only two belt conveyors.
- Short roof spans for buildings.
- Capacity can be increased.

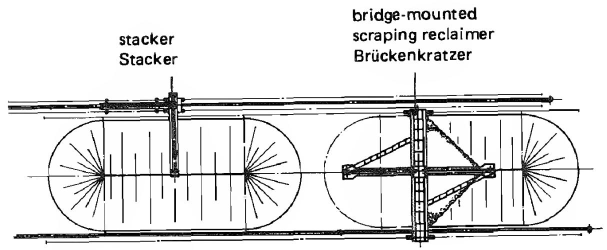

Fig. 37: In-line stockpiles

Disadvantages:

- Long buildings if the stockpiles are under roofed cover.
- End-cone problems.
- High length/width ratio, besides requiring long buildings, makes such blending beds difficult to accommodate in a cement works layout.

3.3.2 Circular stockpile

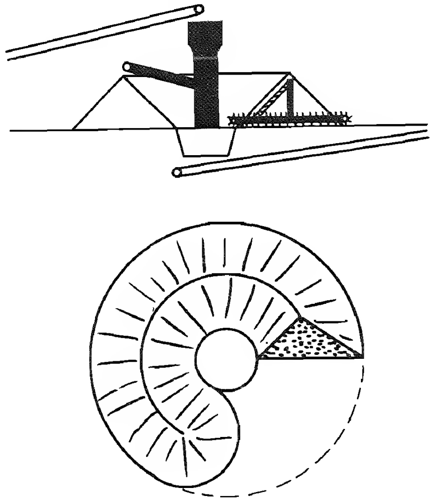

Fig. 38: Circular stockpile

Advantages:

- Very short belt conveyors.
- Simple roof construction for buildings, with central column as supporting member.
- It is relatively simple to keep the reclaiming output rate constant.
- No end-cone problems.
- Area on plan about 40% less than for straight stockpiles.
- No change-over of reclaiming machines.

Disadvantages:

- A circular stockpile is sometimes difficult to fit into the cement works layout.
- Sticky or very moist material may choke the chutes in the central column.
- The poke-holes for unblocking the chutes are relatively inaccessible.
- Ground-water may cause difficulties in the material extraction tunnels.
- Capacity can be increased only by setting up a second pile.

3.3.3 Homogenizing tanks or troughs

Under certain circumstances it may be advantageous to build a sub-surface stockpile, more particularly in a suitably lined excavation formed, for example, by blasting in rock, the object being to save on the cost of above-ground building construction. However, in some cases trough-type blending beds at ground level, i.e., not recessed into the ground, are recommended more particularly by the engineering firm of FLS.

Advantages:

- Very good space utilization.
- No end-cone problems.

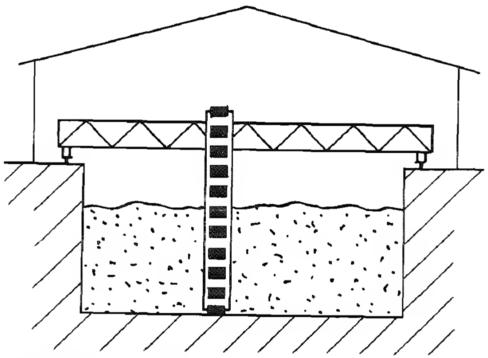

Fig. 39: Homogenizing trough

- Roof can be of simple and light construction
- Very good homogenizing or blending effect.

Disadvantages:

- Trough is expensive to construct.
- Expensive machinery.
- Material falling from a great height throws up much dust.

3.4 Measures to combat end-cone problems

The end cones – i.e., the semi-conical ends – of longitudinal bed-blending stockpiles are liable to cause some problems. For one thing, it is difficult to keep the rate of reclaiming constant at the ends of the pile because here the cross-section of the face from which the material is being reclaimed will vary from slice to slice. Besides, not all the stockpiled layers are then simultaneously removed in each slice. Especially when the reclaimer starts on the stockpile, the homogenizing effect is at first liable to be very unsatisfactory, because there will have been considerable segregation at the time of stacking. Völlmin mentions various methods of counteracting these drawbacks:

- If the capacity of a stockpile is to be increased, it is better to make the pile longer than wider, because the relative volume of the end cones is smaller in a narrower pile.
- For a length/width ratio of 4 the end cones comprise about 15% of the volume of the pile. This proportion increases to about 20% for a ratio of 3.
- The end cone at the "far end" of the pile can be left standing or be only partly reclaimed. This does mean some loss of effective stockpiling capacity, however.
- The reversal points of the stacker can be staggered in relation to the height attained by the pile during the course of building it up. In this way the segregation at the front end can be reduced.
- These end-cone problems are obviated if a circular stockpile is used, especially if the chevcon stacking method is adopted.

4 Sampling stations

For monitoring the operation of single-component or multi-component blending beds which have to attain specified homogenizing or blending effects it is essential to have suitable automatic sampling stations.
So far, not much information on such installations has appeared in the literature, so that guidance on these matters must be sought from the manufacturers of cement plant equipment. There is as yet no standardization of sampling stations, and they are always tailored to suit the requirements of each individual case, which more particularly depend on the properties of the materials to be sampled. This being so, the brief outline presented here can lay no claim to completeness of treatment of the subject.

4.1 Sample quantity

According to available information, a representative sample of raw material in the form of crushed stone will be something between 0.2% and 2% of the total handling flow. If the chemical composition of the material entering the blending bed is subject to large variations, or if it comprises a wide range of particle sizes, sample quantities in excess of 1% should be taken. Stacking rates for modern blending beds are between 400 and 1000 t/hour, so that the sample quantities to be taken and prepared for testing will range from 0.8 to 20 t/hour.
It is advisable to perform the sampling as a weight-dependent rather than as a time-dependent operation. In the former case the sampler is controlled direct by a belt weigher. The cumulative sample is homogenized in a special mixer after a certain quantity (tonnage) of material has been collected. With both methods suitable belt conveyors and automatic counting equipment are required. The samplers can be adjusted to take any desired quantity.
In order to obtain qualitatively correct samples it is necessary to take these from the full cross-section of the flow of material being carried on the raw material belt conveyor.

4.2 Process engineering features

Two raw material sampling systems in actual use at cement works will now be described. In both cases the material in question is limestone.
In general, it is advisable to provide drying facilities, or a crusher that can be heated, for dealing with material with a moisture content of 3% or more. The sampling station is as a rule accommodated in a tower-like structure upstream of the blending bed and comprises the various items of sample preparatory processing machinery installed one above another. By making use of gravity inside the sampling station the material handling equipment can be kept to a minimum and the capital cost and operating expenses of the sampling system be correspondingly reduced. In cases where the sampling of the material can be done only at ground level, it is advisable additionally to install a bucket elevator

4.2.1 Sampling installation 1 (MIAG)

Capacity of raw material handling system:	500 t/hour
Sampling quantity:	1.2% = 0.6 t/hour
Sampling rate:	at 2-minute intervals = 200 kg/cycle
Sample splitter (1:50) ·	4 kg/cycle
Sample mixer:	120 kg charge
Quantity from mixer for despatch to laboratory (1:600 division) ·	200 g/hour.

The samples are taken with a three-compartment chute which intercepts the raw material flow every 2 minutes. The centre compartment diverts the material onto a slow-running belt conveyor which feeds it to a double-shaft hammer crusher which reduces from 30 mm to below 2 mm product size. This crusher is heated, so

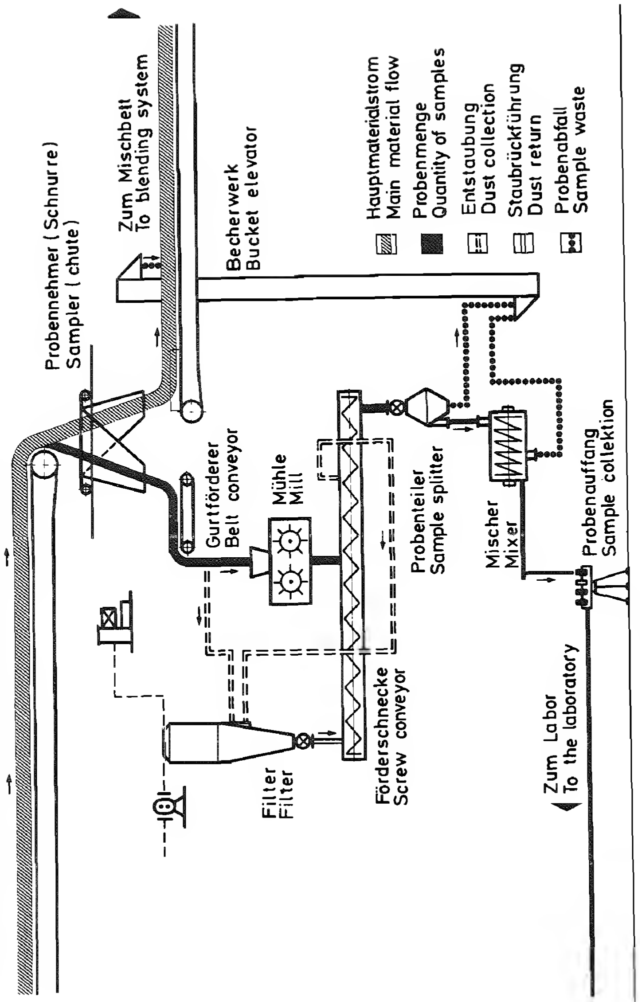

Fig. 40: Flow diagram of sampling station

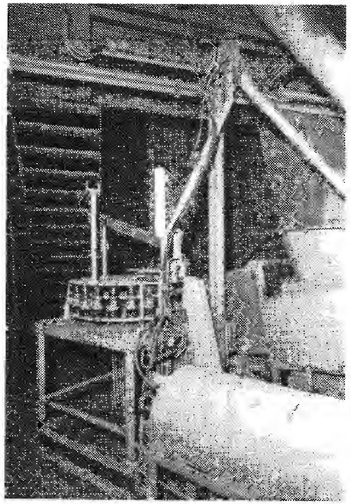

Fig. 41: Sample divider and mixer

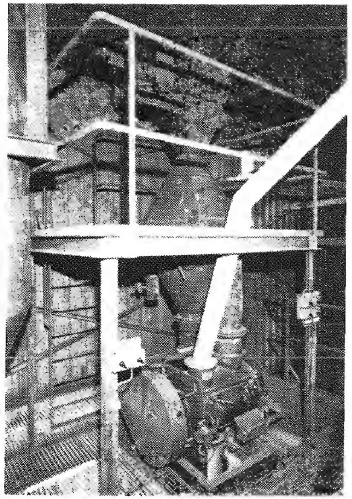

Fig. 42: Turntable in pneumatic despatch station

that the initial 3.5% moisture content is lowered to under 1%. A screw conveyor delivers the sample to a rotary-tube splitter whose discharge opening can be varied from the outside by means of a sliding gate, so that the sample splitting ratio can be adjusted to any desired value. The final reduced sample quantity is collected in a mixer.

A belt weigher incorporated in the belt conveyer bringing the raw material from the quarry records the quantity handled. Under adjustable electronic control, the contents of the mixer are intensively mixed after every 500 t of material passing the weigher. On completion of the mixing operation a quantity of about 200 g is removed from the mixer by a pneumatically powered extraction device and is fed to an automatically functioning pneumatic despatch station which sends the samples to the laboratory. In the laboratory each sample is further dried and prepared for analysis by pulverization in a vibratory mill.

The remainder of the sample material in the mixer is not required for testing and is returned to the main material flow. When the mixer has been emptied, the sampling cycle starts over again. The actual mixing operation is of relatively short duration. No samples are taken during this period, so that there is no risk of the sampling and testing procedure being falsified.

The sampling station has a dust collection system. The dust precipitated in the latter is returned to the sample splitter, so that no dust losses occur.

4.2.2 Sampling installation 2 (FLS)

Capacity of raw material handling system:	500 t/hour
Sample quantity:	0.18% = 10 × 90 kg = 900 kg/hour
1st splitter (1:10):	90 kg/hour
2nd splitter (1:20):	4.5 kg/hour
3rd splitter (1:20):	0.225 kg/hour.

The samples are taken with a swivelling chute which discharges the material onto a vibratory feeder. The latter feeds it to a jaw crusher in which it is reduced from a feed size of up to 50 mm to a product size of about 10 mm. The sampled quantity of material is discharged into the first sample splitter. The reduced quantity is passed through an electrically heated drying device, further crushed to below 1 mm particle size and then further reduced in the second splitter. The sample from this device is crushed for the third time, now to a product size not exceeding 0.2 mm. In the third splitting stage, which then follows, the final sample quantity of about 225 g is obtained.

4.3 Checking the sampling system

Cumulative samples from automatic sampling stations may be affected by systematic errors. The only way to detect such errors is by taking random samples at the same time as the cumulative samples. The random samples are split, prepared

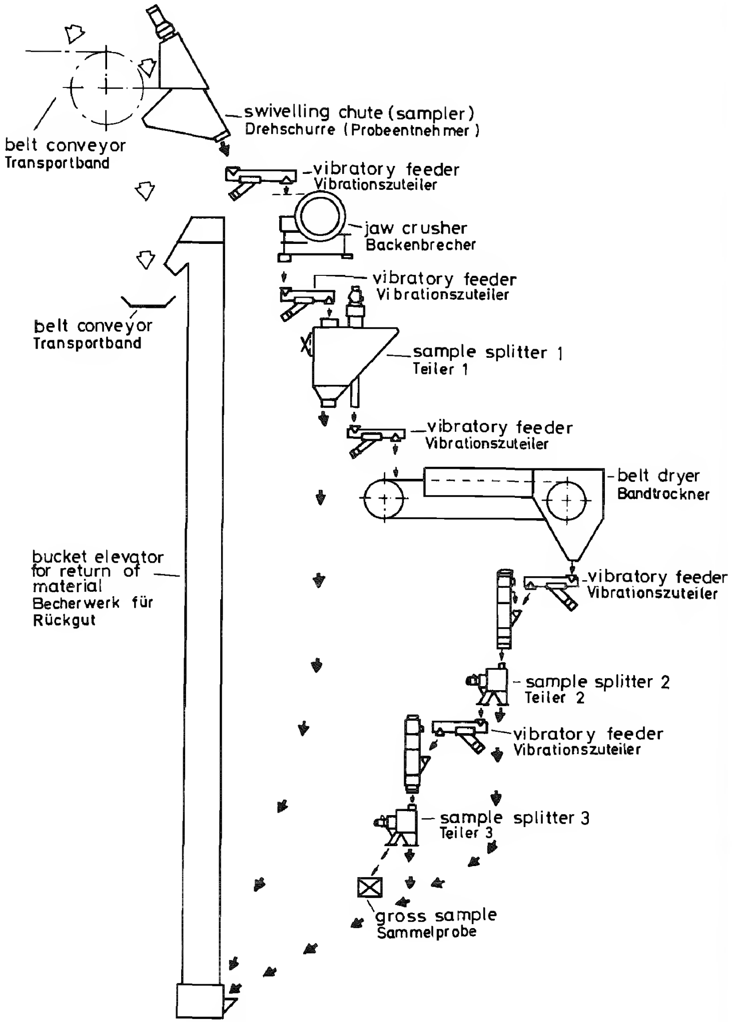

Fig. 43: Automatic sampling station for limestone (FLS)

and analysed by hand. The errors that occur in these operations are greater with increasing maximum particle size of the material from which the smaller subsample has to be obtained by "splitting" the original sampled quantity. Investigations have shown that the error that this may involve in raw material of 0–30 mm particle size range is ± 6.8 lime standard units. On the other hand, the error associated with splitting a sample of comminuted and homogenized material is negligible.

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   d. Holderbank Management u Beratung AG (HMB), – Technische Stelle –, CH-5113 Holderbank (AG)
   e. PHB Pohlig-Heckel-Bleichert, Vereinigte Maschinenfabriken AG, Heckelstr. 1, D-6672 Rohrbach (Saar)
   f F L. Smidth & Co. AS, 77 Vigerslev Alle, DK-2500 Copenhagen-Valby

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Kamm/Zimmer/Frommholz, Pohlig-Heckel-Bleichert (PHB), Rohrbach· Figs.· 4, 12, 16, 17, 29, 30, 31, 36, 37
Schmidt, NORDCEMENT, Hannover: Figs.: 7, 28
Fa. Polysius, Neubeckum: Fig. 32
F. L. Smidth (FLS), Kopenhagen: Fig 43

C. Cement chemistry – cement quality

By D Knöfel

I. Historical introduction

The word "cement" is of ancient Roman origin. The Romans made a kind of structural concrete composed of broken stone or similar material with burned lime as the binding medium. This form of construction was called "opus caementitium". Later on, the term "cementum" was used to denote those admixtures which, on being added to the lime, imparted "hydraulic" properties to it, i.e., gave it the power to set and harden under moist conditions or indeed under water. Such admixtures were more particularly brick dust and volcanic tuff.

The Romans made excellent use of this material. Perhaps their most famous building in which it was employed on a large scale is the Pantheon, a circular temple built in Rome in the reign of the Emperor Hadrian (about 120 A.D.). It is 43 m in diameter and has a domed roof with a circular aperture at the centre. This dome, as well as the walls several metres in thickness, are constructed of "concrete" (the walls are faced with brick). For achieving the hydraulic properties of this concrete the builders used pozzolana, a volcanic tuff from the region of what is now known as Pozzuoli near mount Vesuvius.

Up to the latter half of the 18th century the factors that gave certain types of cementing material their hydraulic properties were shrouded in mystery. The British engineer John Smeaton (1724–1792) recognized the importance of the clay component as essential to hydraulic setting and hardening behaviour when, in 1756, he sought a water-resisting binding medium for the masonry of the new Eddystone lighthouse near Plymouth. More particularly, it was discovered that those cements which did not dissolve completely in nitric acid were found to possess good hydraulic properties (the insoluble residue being due to clay and quartz).

In 1796 another Briton, James Parker, made a hydraulic cement, which he called "Roman cement", from the calcined nodules of argillaceous limestone known as septaria. The first attempts to produce cement by the burning of an artificial mixture of limestone and clay were made in France, especially by Vicat, in the early years of the 19th century. Although these attempts were successful, the results were not followed up in that country, and it was the achievement of Joseph Aspdin, a British bricklayer, to produce an excellent hydraulic lime, in 1824, by burning a mixture containing certain proportions of lime and clay at a high temperature. He called his product "Portland cement", a name which has survived as a generic designation and which was originally chosen by Aspdin because the "artificial stone" made with his cement (and aggregates) was thought to resemble Portland stone, an oolitic limestone found in southern Britain. However, it was not yet a true portland cement as we now know it. This step was achieved by his son William, who succeeded, in 1843, by applying even higher temperatures, to produce a material which contained a substantial proportion of sintered matter in addition to the "underburned" mass of the earlier product. "Sintering" means: burning at a temperature which causes partial fusion of the material. William Aspdin's cement was distinctly superior to its predecessors in attaining higher strengths and was used, inter alia, in building the new Houses of Parliament in London (1840–1852).

The second half of the 19th century saw the rapid expansion of the cement industry in a number of countries, including Germany. The first German cement works, which continued in production for a great many years, was set up at Züllchow, near Stettin, by H. Bleibtreu in 1855, followed by a works at Oberkassel, near Bonn, in 1858. By 1889 there were 60, and around 1900 there were 83 cement works in Germany. The earlier ones used simple intermittently fired shaft kilns. Annular kilns came later. The first rotary kiln in Germany was commissioned in 1898.
In 1862, E. Langen discovered the latently hydraulic properties of granulated (rapidly-cooled glassy) blastfurnace slag, his investigations having shown that mixtures of quicklime and such slag attained high strengths on hardening. The possibility of using portland cement to activate the blastfurnace slag was applied by G. Prüssing in 1882. This principle was, in due course, applied in what in Britain is known as portland blastfurnace cement. In the United States it is known as portland blastfurnace slag cement, while in Germany there are two main varieties, namely, "Eisenportland" cement and "Hochofen" cement. The principle of sulphate activation was discovered by H. Kühl in 1908 and was later to be applied to the manufacture of supersulphated cement. These were mainly German developments. The first high-alumina cements were produced in France during the First World War. Based on patents obtained by J. Bied, a Frenchman, these products consist mainly of the solidified liquid phase (melt) of crystallized monocalcium aluminate.

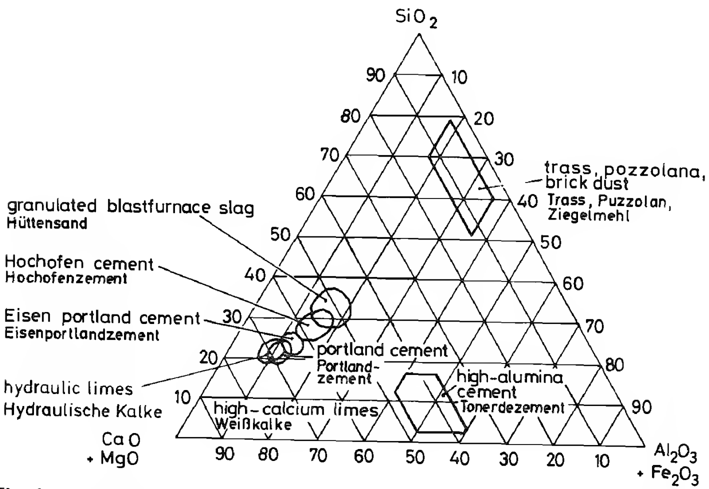


Fig. 1: Diagram of ternary system $SiO_2 - CaO/MgO - Al_2O_3/Fe_2O_3$ (Rankin diagram)

The first set of "Standards for the uniform supply and testing of portland cement" in Germany were issued in 1878. It was the first German Standard relating to a material (and a mass product). The two portland blastfurnace cements, "Eisenportland" and "Hochofen", were standardized in 1909 and 1917 respectively.
The present definition of cement as given in German Standard DIN 1164 is as follows: "Cement is a finely ground hydraulic binding medium for mortar and concrete, consisting substantially of compounds of calcium oxide with silicon dioxide, aluminium oxide and ferric oxide, which have been formed by sintering or fusion. When mixed with water, cement hardens both in air and under water and retains its strength under water; it has to possess constancy of volume (soundness) and attain a compressive strength of at least 25 N/mm^2 at 28 days."
Portland cement is made from portland cement clinker with an admixture of sulphate. Portland blastfurnace cements (slag cements) additionally contain blastfurnace slag, while trass cement additionally contains trass. Besides these cements, other types, such as high-alumina cement and supersulphated cement, are manufactured in some countries, but these two cements are no longer produced in the Federal Republic of Germany and are not standardized here. On the other hand, oil shale cement and trass blastfurnace cement are types which are officially permitted in this country.
The position occupied by cements and allied binding media in the so-called Rankin diagram of the ternary system $SiO_2 - CaO/MgO - Al_2O_3/Fe_2O_3$ is indicated in Fig. 1.
In this chapter the chemical, mineralogical and physical aspects, i. e., the scientific principles, of cement manufacture will be outlined and the corresponding aspects of the application of cement will be briefly dealt with.
The subject will be treated as far as possible in the sequence of the production process: raw materials, preparation of the raw mix, burning and cooling, portland cement clinker, grinding, storage, types of cement. Various tests applicable to cement will be described. In addition, since it is essential for the cement manufacturing engineer to know something also of the practical application of his product (e. g., in connection with testing and in dealings with customers), the phenomena associated with the hardening (hydration) of cement will also be considered.

II. Raw materials and the raw mix

1 Raw materials

1.1 General considerations; origins

The ideal raw material for cement manufacture is a rock which already in its natural state contains the correct proportions of the constituents to produce a cement clinker of the desired composition. Besides, it should be available in abundance, easy to quarry and of homogeneous character. In reality this ideal combination is extremely rare. Instead, it is nearly always necessary to base the manufacture of

cement on raw materials which are not in themselves very suitable, but which have to be appropriately combined and blended. For practical purposes the raw materials are limestone and clay (occurring in deposits in which they are usually mixed with certain amounts of other components).
Limestones and clays are, in the geological sense, sedimentary deposits. These may be formed inorganically from the weathering residues or the precipitated solution products of older rocks (e.g., granite or basalt, but also sandstone, limestone and marble) or may occur as new formations. The latter may be inorganic in character (e.g., clays formed from weathering products) or organic (e.g., chalk formed from the shells of marine organisms). Most sedimentary deposits are of marine origin, i.e., formed in seas (most limestones, for example). Clays are deposited in lakes, along rivers and as offshore formations in seas. Some sediments subsequently undergo processes of change and consolidation (diagenesis) (Fig. 2).
The typical form in which sediments are laid down is in layers, known as strata or beds in geological terminology. Since they are nearly always deposited in water, the layers are originally horizontal. The actual stratification, i.e., the presence of individually distinguishable layers, is caused by variations in the sedimentation or other conditions governing the formation of the sediment. As a result of these geological processes over millions of years, deposits of considerable depth (thickness), sometimes amounting to hundreds of metres, may be built up. Though originally horizontal and extending uniformly over large areas, these strata may subsequently be affected by so-called tectonic processes – upheavals and disturbances of various kinds – which cause them to become tilted, folded, faulted or disrupted in other ways. When such deposits are quarried, their stratified

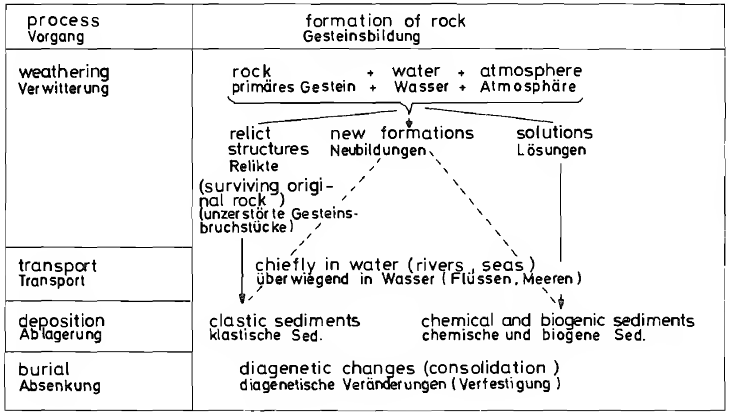


Fig. 2: Diagram of sediment formation

character is usually quite clearly discernible, but the strata may be discontinuous, displaying sudden breaks which must be taken into account in quarrying the material.
Limestones consist predominantly of calcium carbonate ($CaCO_3$), generally in its most stable modification known as calcite. In addition, they often contain magnesium, aluminium and iron combined as carbonates and silicates; silica (SiO_2), usually in the form of quartz, is also often present. Most limestones utilized by the cement industry are either chemically precipitated or organic limestones.
Chemically precipitated limestones are formed more particularly in warm seas where water supersaturated with lime and of low CO_2 content may occur (e.g., at present on the Bahama banks). This inorganic process of precipitation proceeds as follows

$Ca(HCO_3)_2$	$\rightarrow$ $CaCO_3$ +	H_2O +	CO_2
dissolved in sea water	precipitation of lime	water	given off as gas.

A fairly common variety of limestones in this category are the oolitic limestones, which are composed of so-called ooliths, i.e., more or less spherical rock particles grown by accretion around a nucleus and of the order of 1 mm in diameter. These calcareous ooliths are formed in shallow water (less than about 2 m depth) subject to considerable motion. When a certain amount of lime has been deposited around the nucleus (which may be a grain of sand or a shell fragment), the oolith sinks to the bottom by gravity. Portland limestone belongs to this type.
The organic, or biogenic, limestones represent a substantial proportion of limestones. Many marine organisms – plants and animals – form hard shells or skeletons of calcium carbonate. When they die, their calcareous remains accumulate as a sedimentary deposit. Such organisms are, for example, many species of algae, corals, shellfish and protozoa (more particularly the Foraminifera). If these are distinctly identifiable in the limestone as fossil remains, they form the basis of classification, e.g., shelly limestones, coral limestones, algal limestones, foraminiferal limestones, etc. Chalk is a limestone consisting mainly of the remains of unicellular planktonic algae, more particularly so-called coccoliths, which are microscopic calcareous plates secreted by those organisms.
Marble is a limestone consisting very largely of calcite ($CaCO_3$) in a relatively coarsely crystalline form. It is what is known as a metamorphosed limestone produced under conditions of high temperature and pressure, more particularly in the process of mountain formation (orogenesis). On account of its hardness, marble is seldom used as a raw material for cement.
There are various transitional types and varieties of limestone.
Clays are clastic sediments, i.e., they consist mainly of the remains of pre-existing rocks which have been broken down by weathering and/or erosion. The clay minerals are present in the form of very small particles (<0.002 mm) which have been deposited mainly in water – fresh, brackish or marine. Geologically the clays, along with shales, marls, etc., are classed as argillaceous rocks. The term "clay" is more especially reserved for material which has no pronounced bedding planes and which forms a plastic mass when wet. The principal constituents are the clay

minerals, and it is these that give the clay its plastic properties. They are so-called layer-lattice minerals and occur mostly in the form of minute plate-like, occasionally fibrous, crystals. Most clays consist of different clay minerals which are present together, e.g., illite, montmorillonite, kaolinite, halloysite, etc. Their chemical composition is far from simple, as the following two examples will show:

montmorillonite	$Al_2[(OH)_2\ Si_4O_{10}] \cdot 4H_2O$
kaolinite	$Al_4[(OH)_8\ Si_4O_{10}]$.

Besides clay minerals, clays may contain various proportions of other finely divided substances: quartz (SiO_2 "sand"), calcite ($CaCO_3$), gypsum ($CaSO_4 \cdot 2H_2O$), limonite (FeOOH), pyrite (FeS_2), feldspars (aluminosilicates), carbonaceous particles, etc.
Clay soils with a substantial proportion of sand and silt, and often with a certain amount of limonite (iron oxides and hydroxides giving the material a yellowish or brownish colour), are called **loam**. The term **marl** is applied to calcareous mudstones, which are natural mixtures of clay and lime. **Loess** is formed as an accumulation of wind-born dust with particles in the size range of 0.01 to 0.1 mm, originally derived from desert areas and of a brownish-yellow colour. The constituents are mainly siliceous (clay, quartz, feldspar) and about 10% of lime. If the lime has been dissolved out, the material is called loess loam.

1.2 Use in cement production

As a rule, the main components available for the manufacture of cement are limestones (the source of CaO) and clays (the source of SiO_2, Al_2O_3 and Fe_2O_3). These have to be mixed with each other in proportions depending on their own and on the required final chemical composition. However, overall chemical composition is not the deciding factor, because the reactions in the cement burning process take place between the individual phases present in the kiln feed mix, so that the fineness and homogeneity of the raw material and raw meal are also important. If the kiln feed has a large reactive surface area and the mineral phases are homogeneously distributed, the diffusion rates and therefore solid reaction velocities will be higher than in coarser and less well homogenized material. The reaction behaviour of those raw materials whose natural composition is already fairly close to the desired chemical composition (lime marl, for instance) will generally be more favourable, because the components are naturally present in a very finely crystalline and well blended form. On the other hand, mixtures of "extreme" raw materials (e.g., pure limestone and pure clay) react less favourably.
As already stated, the CaO-bearing component is usually a limestone. Limestones which already contain some natural admixture of clay are to be preferred, as already noted above. The following approximate classification is applicable:

pure limestone	$>95\%\ CaCO_3$ (by weight)
marly limestone	$85-95\%\ CaCO_3$ (by weight)
lime marl	$70-85\%\ CaCO_3$ (by weight)
marl	$30-70\%\ CaCO_3$ (by weight)
clay marl	$15-30\%\ CaCO_3$ (by weight)
marly clay	$5-15\%\ CaCO_3$ (by weight)
clay	$<5\%\ CaCO_3$ (by weight).

The raw materials for cement manufacture have a $CaCO_3$ content between about 74 and 79% by weight.
Some limestones contain a certain amount of dolomite $CaMg(CO_3)_2$ and thus introduce magnesium oxide (MgO) into the raw material. Magnesia expansion must be reckoned with if the MgO content exceeds about 5% by weight.
The oxides SiO_2, Al_2O_3 and Fe_2O_3 are generally provided by an argillaceous component, i.e., a clay or allied material (clay, marly clay, clay marl). Raw materials containing sand are sometimes also used, e.g., sandy marl or sandy limestone. In some instances these components may contain harmful concentrations of alkalies (K_2O, Na_2O), sulphates (e.g., gypsum $CaSO_4 \cdot 2H_2O$; the sulphates are usually reckoned as SO_3) and, more rarely, chlorides. These substances may cause difficulties in the burning process, more particularly in consequence of intensified cyclic processes and coating formation in the kiln system. The clays also have a major effect on the pelletizing or nodulizing properties of the raw meal and on the water demand of the raw slurry in the wet process of cement manufacture.
If it is not possible to obtain the desired chemical composition of the raw mix just with the two above-mentioned raw material components, it will be necessary to add relatively small quantities of **corrective ingredients** to the mix. These should contain the required oxides – deficient in the main raw materials – in fairly high concentrations. At the same time, however, they must not contain appreciable amounts of harmful oxides (e.g., MgO or K_2O). Their purpose, therefore, is to adjust the chemical composition of the raw mix and improve its sintering capacity. More particularly, the following are used: **quartz sand** for increasing the silica content; **roasted pyrites** or **iron ore** for increasing the ferric oxide content (these substances should contain at least 25% Fe_2O_3).
Other corrective ingredients are sometimes used, depending on local availability and need.
Blastfurnace slag is only exceptionally used as a raw material component for cement manufacture (it is, however, extensively used as a subsequent additive to cement in the production of portland blastfurnace cement).
If solid fuels are used in the burning process, the **ash** arising from these will become incorporated in the cement and have to be taken into account.

2 Raw mix: proportioning and analysis

2.1 Principles of proportioning the raw materials

For the production of cement it is necessary to have, or make, raw material mixtures whose chemical composition is within certain limits. The continuous production of high-quality cement is possible only if the raw mix possesses optimum com-

Table 1: Limiting values of chemical composition of cement raw material (after ignition)

oxide	limiting value [M.-%]	content [M.-%]
CaO	60–69	65
SiO_2	18–24	21
Al_2O_3	4– 8	6
Fe_2O_3	1– 8	3
MgO	<5,0	2
K_2O, Na_2O	<2,0	1
SO_3	<3,0	1

position and furthermore if variations in this composition remain within the narrowest possible range. The limiting values stated in Table 1 are to be regarded as valied for the manufacture of cement generally, i.e., they relate to all manner of cement works. Within any particular works the variations have to be much smaller.
For practical purposes the raw material composition (and also the composition of the cement clinker) is usually characterized by certain ratios, often called "moduli". They are in fact proportioning formulas into which the percentages of the various oxides, as determined by chemical analysis, should be substituted.
For calculating the optimum lime content of the mix, the so-called **hydraulic modulus**, as expressed by the following formula, may be used:

$$HM = \frac{CaO}{SiO_2 + Al_2O_3 + Fe_2O_3}.$$

Nowadays, however, this has largely been superseded by the **lime standard** (LSt), for which some variant formulas have been evolved.
A high content of lime (CaO) enables lime-rich clinker phases, which have the most favourable properties (especially with regard to strength development), to be formed more abundantly in the burning process, but subject to the condition that all the CaO must be combined with the three other major oxide components (SiO_2, Al_2O_3, Fe_2O_3). The object of the proportioning formulas is to provide a means of calculating the maximum proportion of lime that can be made to combine with these acidic oxides.
If there is an excess of uncombined lime, i.e., existing as free lime (CaO_{fr}) in the cement, it may cause damage in mortar or concrete as a result of expansion phenomena (see also Section IV.1). The lime standard provides a criterion for determining the optimum lime content. It expresses the actual content of CaO present in the raw material (or in the clinker) as a percentage of the maximum CaO content which can be combined by the acidic oxides (SiO_2, Al_2O_3, Fe_2O_3) in the most lime-rich clinker phases under technical conditions of burning and cooling.

Thus LSt = 100 represents the optimum CaO content. Two formulas for the lime standard will be given here. The first, designated LSt I, was due to Kühl:

$$LSt\,I = \frac{100\,CaO}{2.8\,SiO_2 + 1.1\,Al_2O_3 + 0.7\,Fe_2O_3}.$$

A somewhat modified version was later substituted as LSt II, while LSt III, due to Spohn, Woermann and Knöfel, furthermore takes account of the possible presence of MgO, which can replace up to 2% (by weight) of CaO:

$$LSt\,III = \frac{100\,(CaO + 0.75\,MgO)}{2.80\,SiO_2 + 1.18\,Al_2O_3 + 0.65\,Fe_2O_3}.$$

Values of the MgO content only up to 2% are to be taken into account in the LSt III formula; if this content is higher, the second term in parentheses should remain constant at 1.50.
As a further refinement, the term $-0.7\,SO_3$ may be introduced into the numerator, to take account of the possible formation of $CaSO_4$. This is done, for example, in the generally similar "lime saturation factor" used in British cement manufacturing practice.
Example of the application of the lime standard:
The chemical analysis of a raw meal gives the following results (in % by mass or weight).
65.7 CaO, 21.1 SiO_2, 6.6 Al_2O_3, 3.1 Fe_2O_3, 2.0 MgO, residue 1.5;

$$LSt\,III = \frac{100\,(65.7 + 75 \times 2.0)}{2.80 \times 21.1 + 1.18 \times 6.6 + 0.65 \times 3.1} = 97.5.$$

For technical clinkers the value of LSt III is between 90 and 102 (values above 97 are to rated as very high-quality).
The **silica modulus** (SM) (or silica ratio) is the ratio of silica (SiO_2) to the sum of the alumina (Al_2O_3) and ferric oxide (Fe_2O_3).

$$SM = \frac{SiO_2}{Al_2O_3 + Fe_2O_3}.$$

This modulus characterizes the ratio of solid to liquid in the clinkering of the material, because at clinkering temperature the SiO_2 is predominantly present in the solid phases (alite and belite), whereas the other two oxides occur in the liquid phase (melt). In industrial cements the silica modulus is generally between 1.8 and 3.0.

The **iron modulus** (IM), also known as the **alumina** ratio (AR), is the ratio of alumina to ferric oxide:

$$IM = \frac{Al_2O_3}{Fe_2O_3}.$$

Since these two oxides both occur almost entirely in the liquid phase at clinkering temperature, this modulus characterizes the composition of that phase. If the ferric oxide content is higher, so that the iron modulus is lower, the viscosity of the melt decreases. For a value of $IM < 0.638$ the clinker phase called tricalciumaluminate (C_3A) fails to form: C_3A-free cements are characterized by increased sulphate resistance. In industrial cements this modulus is generally between 1.3 and 4.0 and most often between 1.8 and 2.8. In special cements it may have much lower values (down to about 0.4).

In the burning process, volatile constituents are driven out of the raw materials. More particularly, carbon dioxide (CO_2) is driven out of the limestone and water of hydration is driven out of the clay. As a result, the materials undergo a decrease in weight in the production of cement clinker.

The required quantity of dry raw material (i.e., without its inherent natural moisture) for the production of portland cement clinker can be computed as follows:

The carbon dioxide is driven out of the limestone:

$$\begin{array}{l} Ca \quad C \quad O_3 = Ca \quad O + C \quad O_2 \\ (40 + 12 + 48) = (40 + 16) + (12 + 32) \end{array}$$

100 parts $CaCO_3$ = 56 parts CaO + 44 parts CO_2.

Furthermore, about 7% of water of hydration is expelled from the clay in the raw meal (organic constituents, etc. are not considered).

Thus, when a raw meal containing, say, 76% $CaCO_3$ and consisting only of $CaCO_3$ and clay is ignited, the loss on ignition will be approximately as follows:

$$\begin{array}{ll} \text{from } CaCO_3 & = 0.76 \times 44 = 33.44\%\ CO_2 \\ \text{from clay} & = 0.24 \times 7 = 1.68\%\ H_2O \\ \hline & \ 35.12\% \text{ total loss on ingnition,} \end{array}$$

i.e., raw meal with 76% $CaCO_3$ gives about 64.9% of clinker, for an ignition loss of about 35.1% (or: 1 kg of raw meal yields about 0.65 kg of clinker)

For different values of the $CaCO_3$ of the raw meal the quantities of materials can be calculated with the following formulas:

$$a = 1 - \left(\frac{0.44 \times \%\ CaCO_3}{100} + \frac{0.07 \times (100 - \%CaCO_3)}{100}\right) = \text{kg of clinker per kg of raw meal}$$

$$b = \frac{1}{a} = \text{kg of raw meal per kg of clinker}$$

$$c = \frac{\%\ CaCO_3 \times 56}{a \times 100} = \%CaO \text{ in the clinker.}$$

The following values are obtained with these formulas:

% $CaCO_3$ in the raw meal	74	75	76	77	78	79
a = kg of clinker per kg of raw meal	0.656	0.652	0.649	0.645	0.641	0.638
b = kg raw meal per kg of clinker	1.524	1.533	1.541	1.550	1.558	1.567
c = % CaO in the clinker	63.2	64.4	65.6	66.9	68.1	69.3

Intermediate values can be directly read from the accompanying diagrams (Fig. 3).

Taking account of losses of material in the manufacturing process, it is generally assumed in practice that 1.55–1.60 kg of raw material is needed for producing 1 kg of clinker.

2.2 Calculation of the raw mix proportions

(a) For the **approximate calculation** of the mix proportions for two raw material components it is convenient to set down the relevant values in an "X" pattern, at the centre of which is written the desired $CaCO_3$ content of the raw mix. The $CaCO_3$ content of the limestone is written in the upper left-hand corner, and the $CaCO_3$ content of the clay is written in the lower left-hand corner. The differences between the two last-mentioned values and the desired $CaCO_3$ content of the raw mix at the centre of the "X" are now written in the diagonally opposite corners. The values thus finally obtained represent the proportions of the raw materials which will form the desired mix.

Example:

Suppose that the following raw materials are available:

%	SiO_2	Al_2O_3	FeO_3	CaO	MgO	loss on ignition
limestone	3.8	0.9	0.6	52.9	0.3	41.5
clay	53.4	20.2	7.5	4.3	2.1	12.5

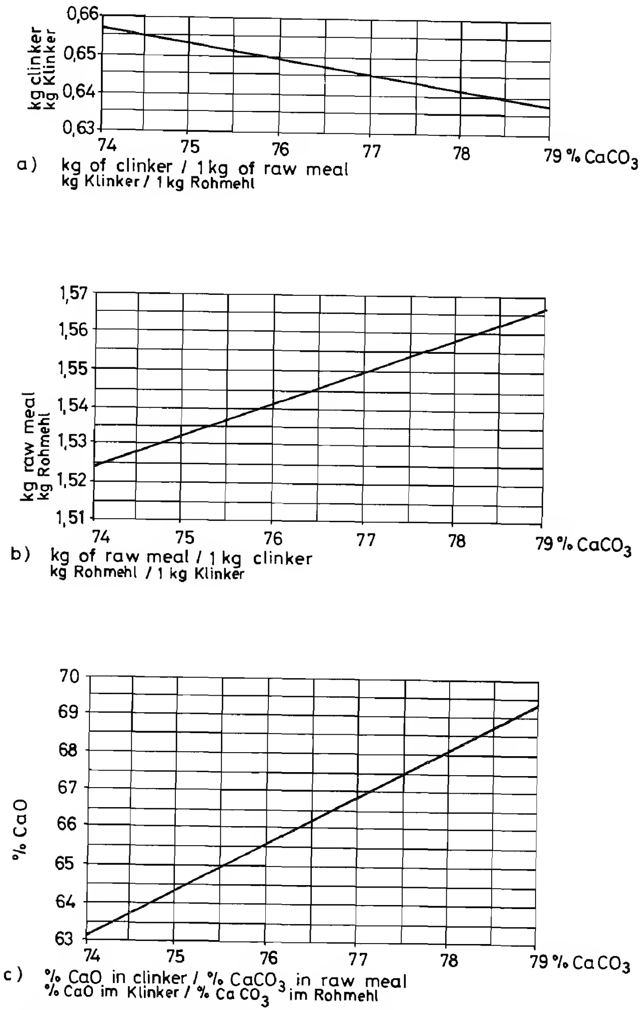


Fig. 3: Yield obtained for different percentages of $CaCO_3$ in the raw meal (from Labahn/Kaminsky, 1974)

The limestone contains $\dfrac{52.9 \times 100}{56} = 94.5\%\ CaCO_3$.

The clay contains $\dfrac{4.3 \times 100}{56} = 7.7\%\ CaCO_3$.

(It has been assumed that all the CaO is present as $CaCO_3$.)
For 77% $CaCO_3$ in the raw meal the above-mentioned "X" pattern for computation gives:

94.5 ↘ ↗ 69.3 (parts of $CaCO_3$ deficient in the clay)

77

7.7 ↗ ↘ 17.5 (parts of $CaCO_3$ in excess in the limestone).

The raw mix should therefore be proportioned as follows:

$$\frac{\text{limestone}}{\text{clay}} = \frac{68.3}{17.5} = \frac{3.96}{1}.$$

The following analysis values are calculated:

	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	loss on ignition
limestone (3.96 parts)	15.1	3.6	2.4	209.5	1.2	164.3
clay (1 part)	53.4	20.2	7.5	4.2	2.1	12.5
	68.5	23.8	9.9	213.8	3.3	176.8 (496.1)
raw mix (%)	13.8	4.8	2.0	43.1	0.7	35.6 (100)
raw mix, ignited (%)	21.4	7.5	3.1	66.9	1.1	–

It is further necessary to check that the composition of the raw mix calculated in this way is within the permissible limits (Table 1) and to ascertain its lime standard (more particularly LSt III, which in this case is 95.7) and moduli (in this case: silica modulus = 2.0, iron modulus = 2.4). If necessary, a different lime content will have to be chosen or corrective materials added.
(b) Calculation of a two-component raw mix with the aid of **Kühl's lime standard formula** [41]:

Suppose that the available components are limestone (k) and clay (t) with the following composition (in % by weight):

	limestone k	clay t
SiO_2	$S_k = 5.0$	$S_t = 57.6$
Al_2O_3	$A_k = 1.9$	$A_t = 25.4$
Fe_2O_3	$F_k = 1.4$	$F_t = 9.7$
CaO	$C_k = 91.2$	$C_t = 4.9.$

What quantity x of clay t must be mixed with 1 part of limestone k to obtain the raw mix for portland cement with the lime standard LSt?

In general, the raw mix composed of 1 part of limestone k and x parts of clay t will have the following composition:

$$SiO_2 = S_k + x \cdot S_t \qquad Al_2O_3 = A_k + x \cdot A_t$$
$$Fe_2O_3 = F_k + x \cdot F_t \qquad CaO = C_k + x \cdot C_t.$$

According to the formula for the lime standard LSt I (see Section II.2.1):

$$\text{LSt I} = \frac{100 \cdot (C_k + x \cdot C_t)}{2.8\ (S_k + x \cdot S_t) + 1.1\ (A_k + x \cdot A_t) + 0.7\ (F_k + x \cdot F_t)}.$$

On solving this equation for x, we obtain:

$$x = \frac{\text{LSt I} \cdot (2.8 \cdot S_k + 1.1 \cdot A_k + 0.7 \cdot F_k) - 100 \cdot C_k}{\text{LSt I} \cdot (2.8 \cdot S_t + 1.1 \cdot A_t + 0.7 \cdot F_t) - 100 \cdot c_t}.$$

On substitution of the oxide formulas this becomes:

$$x = \frac{\text{LSt} \cdot \text{I}\ (2.8 \cdot SiO_2 + 1.1 \cdot Al_2O_3 + 0.7 \cdot Fe_2O_3 - 100\,CaO\ \text{(for limestone)}}{\text{LSt} \cdot \text{I}\ (2.8 \cdot SiO_2 + 1.1 \cdot Al_2O_3 + 0.7 \cdot Fe_2O_3 - 100\,CaO\ \text{(for clay)}}.$$

For LSt I = 98 and the above-mentioned oxide concentrations we can calculate: x = 0.400.

Therefore in this case the raw mix must consist of 1 part of limestone and 0.400 part of clay. The precise composition of the mix is as follows:

	SiO_2	Al_2O_3	Fe_2O_3	CaO
1 part limestone	5.0	1.9	1.4	91.2
0.400 part clay	23.0	10.1	3.9	2.0
	28.0	12.0	5.3	93.2

On the assumption that the clinker made from this raw mix contains 5% (by weight) of constituents unaccounted for (loss on ignition, MgO, alkali oxides, etc.), the following clinker analysis is calculated:

SiO_2	= 19.2% by weight	LSt	= 98
Al_2O_3	= 8.2% by weight	SM	= 1.6
Fe_2O_3	= 3.6% by weight	IM	= 2.3
CaO	= 63.9% by weight.		

Since the SM in this example is very low (cf. information given in Section II.2.1), it would be advantageous to add an appropriate quantity of quartz sand as a third raw material component (corrective material).

For the calculation of a three-component raw mix see, for example, Kühl, 1963, p. 99ff. Formulas for the calculation of a mix comprising four components are given by Seidel/Huckauf/Stark, 1978, p. 61 ff.

The calculation of raw mix proportions is useful in connection with the planning of new cement works or the opening-up of new raw material deposits in that it provides approximate guidance on the quantities of materials required and on the suitability of the available raw material components. On the other hand, such calculations are of dubious value for routine production purposes, because in practice the respective components are continually subject to more or less substantial variations. This would necessitate regular analytical monitoring of the raw materials and, on the basis of the results, continual recalculation of the mix proportions.

2.3 Raw mix (or raw meal) analysis

For the purpose of production control the raw mix or the raw meal and the clinker are regularly analysed. Besides "wet" chemical analysis, X-ray analysis is extensively used for the purpose in modern cement works.

The "non-destructive" X-ray-based analytical techniques have now been in widespread and successful use in the cement industry for about 15 years and are employed either for quantitative elemental analysis (X-ray fluorescence analysis, X-ray spectrometry) or for quantitative phase determination (X-ray diffraction analysis, X-ray diffractometry). Of course, purely qualitative checks can also be made by these methods. Whereas X-ray fluorescence analysis (elemental determination) is extensively used both for raw meal and for cement monitoring, X-ray diffractometry (phase analysis) has hitherto been used only for the determination of free lime content of cement.

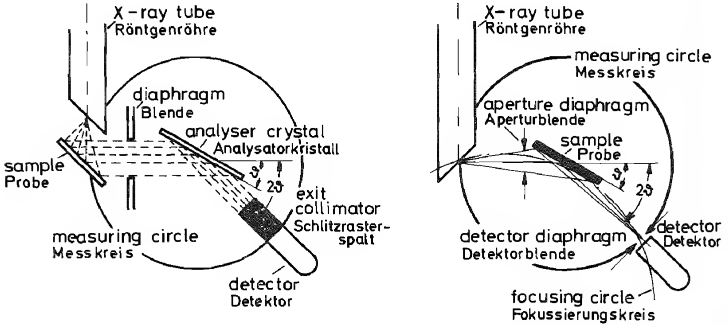
In the cement industry more particularly the so-called wavelength-dispersive principle of X-ray fluorescence analysis is generally used (in preference to the energy-dispersive principle) because it achieves higher intensities and better resolution, so that errors are smaller. The methods of sample preparation are also important deciding factors with regard to the reliability of the X-ray analysis results.

The physical basis for X-ray analysis is the equation commonly known as Bragg's law:

$$n \cdot \lambda = 2d \cdot \sin \vartheta$$

where: λ = wavelength of the radiation
ϑ = angle of incidence and diffraction angle
d = spacing of crystal lattice planes
n = integer.

In X-ray fluorescence analysis the values of d and ϑ are known by virtue of the instrumentation set-up, while λ, the characteristic wavelength of the emitted radiation, is determined. On the other hand, with X-ray diffractometry the values of λ and ϑ are known, while d, the characteristic lattice spacing of a crystalline material, is determined.
In the case of fluorescence analysis the sample is irradiated with high-energy X-rays in the spectrometer. The radiation dislodges electrons from the "inner shells" of the atoms, and the vacant positions are immediately occupied by electrons from the "outer shells" These last-mentioned electrons thus pass into a lower-energy state, and the accompanying release of energy is emitted as X-rays (of various wavelengths) which are typical of each type of atom, i.e., each chemical element. The intensity of this emitted characteristic X-radiation is measured and is proportional to the quantities of the respective elements present in the sample under investigation.
It emerges from Bragg's law that the characteristic X-rays are to be measured at certain values of the angle between the sample and the detector. There are



ray path in spectrometry
Strahlengang bei Spektralanalyse

ray path in diffractometry
Strahlengang bei Beugungsanalyse

Fig. 3a: X-ray analysis methods

two systems of measurement: either the detection channel (comprising analyser crystal and detector) is moved through a certain angular range and measures the characteristic radiation of the elements successively (sequential system) or the apparatus is equipped with a number of detection channels in a fixed array, namely, one channel per element to be detected (multichannel simultaneous system). The sequential system offers greater flexibility, enabling a variety of elements to be detected. On the other hand, the simultaneous system is quicker, comprises fewer moving parts and has proved advantageous more particularly in cases where the same elements have to be analysed over and over again in the routine samples. Hence this last-mentioned system is preferred for production control use in the cement industry. With such equipment a complete analysis can be obtained in a few minutes (the actual analysis and measuring time is very short, e.g., 20 seconds).
X-ray fluorescence spectrometers can be used independently for occasional analyses or for simple analysis programs or be incorporated as an integral part of a process control system associated with computer equipment. The output data may take the form of pulse rates, concentrations (% by weight), moduli and lime standard, or be utilized in some other form for process control at the raw material end of the cement manufacturing process (e.g., for raw mix proportioning feed control).

References

3, 4, 5, 17, 22, 23, 29, 30, 46, 60, 64, 65, 66, 70, 77, 87

III. Chemical, physical and mineralogical aspects of the cement burning process

For the production of cement clinker the raw material has to be heated to a temperature of about 1450° C, so that clinkering occurs. The burning process requires an oxidizing atmosphere in the kiln, producing a greyish-green clinker If this condition is not satisfied, the resulting clinker will be of a brown colour, and the cement obtained from it will be of inferior strength and will set more rapidly. Important chemico-physical processes occur already during the heating-up of the kiln feed material and especially at the burning temperature (clinkering temperature), such as: dehydration of the clay minerals, decarbonation (expulsion of carbon dioxide) of the carbonates (this process is usually referred to as calcining or calcination), solid reactions and reactions involving the participation of a liquid phase (melt), and crystallization processes. All these processes are significantly affected not only by chemical factors (chemical composition of the raw materials),

but also by mineralogical (mineral composition) and physical factors (particle size, homogeneity, etc.). The due completion of these endothermic reactions plays a decisive part with regard to the quality of the cement produced. Table 2 reviews the transformations in the processing of the raw meal; these will be discussed below. Fig. 4 and 5 give information on the formation of new phases that occurs in the kiln system.

Table 2: Chemical transformations in the thermal treatment of portland cement raw meal (principal reactions in clinker burning)

temperature °C	process	chemical transformation
< 200	escape of free water (drying	
100 . . 400	escape of adsorbed water	
400 . . 750	decomposition of clay, e. g., with formation of metakaolinite	$Al_4(OH)_8Si_4O_{10}$ $\rightarrow 2(Al_2O_3 \cdot 2SiO_2) + 4H_2O$
600 .. 900	decomposition of metakaolinite and other compounds, with formation of a reactive oxide mixture	$Al_2O_3\ 2SiO_2 \rightarrow Al_2O_3 + 2SiO_2$
600 ... 1000	decomposition of limestone, with information of CS and CA	$CaCO_3 \rightarrow CaO + CO_2$ $3CaO + 2SiO_2 + Al_2O_3$ $\rightarrow 2(CaO \cdot SiO_2) + CaO\ Al_2O_3$
800 .. 1300	uptake of lime by CS and CA, formation of C_4AF	$CS + C \rightarrow C_2S$ * $2C + S \rightarrow C_2S$ $CA + 2C \rightarrow C_3A$ $CA + 3C + F \rightarrow C_4AF$
1250 . 1450	further uptake of lime by C_2S	$C_2S + C \rightarrow C_3S$

1 Drying

The water that is present as "free" (uncombined) moisture in the raw meal, or has been added to it (e. g., for pelletizing), is driven out at temperatures ranging up to about 200° C.

* For abbreviated notation see footnote on page 123

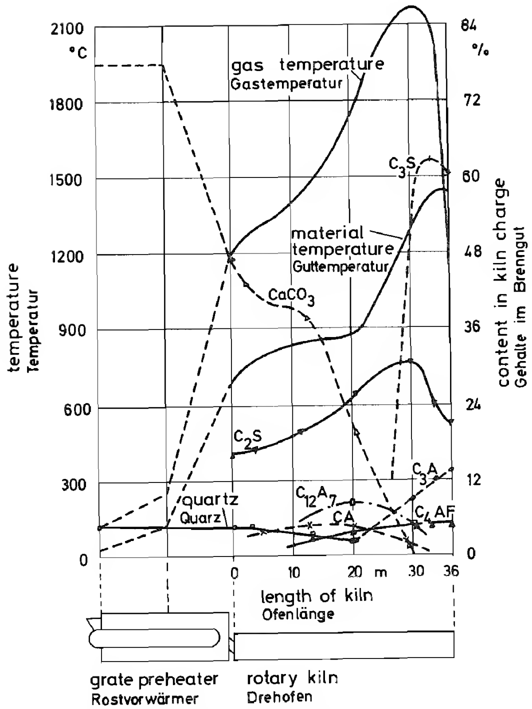


Fig. 4: Formation of new phases in the Lepol kiln (from Weber, 1960)

2 Dehydration of clay minerals

Between about 100° and 400° C the clay minerals give off their adsorptively bound water, including the so-called interlayer water. At higher temperatures, depending on the types of clay mineral concerned, generally between about 400° and 750° C, the chemically combined water (hydroxide groups) is also expelled (dehydration), exemplified by the dehydration of kaolinite·

$$\underset{\text{kaolinite}}{Al_4[(OH)_8Si_4O_{10}]} \rightarrow \underset{\text{metakaolin}}{2(Al_2O_3 \cdot 2SiO_2)} + 4H_2O$$

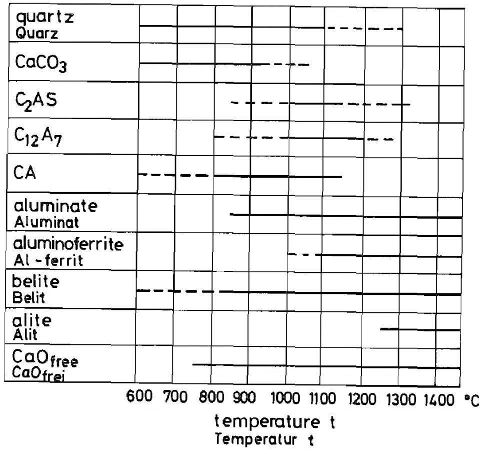


Fig. 5: Existence ranges of the phases in the charge (phase determination in cooled samples; — confirmed information, - - - - reported only by some authors (from Seidel/Huckauf/Stark, 1978)

Metakaolin undergoes decomposition already to some extent within the above-mentioned temperature range and further up to about 900° C, resulting in the formation of reactive oxides, e. g., as follows. $Al_2O_3\ 2SiO_2 \rightarrow Al_2O_3 + 2SiO_2$

The dehydration of clays is affected by various factors, such as the type of clay mineral, the nature and quantity of admixtures, the particle size, the degree of crystallization of the clays, the gaseous atmosphere, etc.

3 Decomposition of carbonates

The calcium carbonate ($CaCO_3$) which constitutes about 74 to 79% of the cement raw meal is decomposed (dissociated, decarbonated, calcined) at temperatures from, theoretically, 896° C upwards, in accordance with the equation:

$CaCO_3 \rightarrow CaO + CO_2$.

At that temperature the dissociation pressure is > 1 bar and thus equals the external pressure. The requisite reaction enthalpy ΔH is 1660 kJ/kg. The value of 896° C relates to pure calcite; with increasing content of admixtures (e. g., in cement raw meal) the thermal dissociation shifts to lower temperatures. In actual practice it begins between 550° and 600° C. This effect is due to chemical reactions of the CaO with the admixtures SiO_2, Al_2O_3 and Fe_2O_3, resulting in the formation

initially of, for example, $CaO \cdot Al_2O_3$ (= CA), $12CaO \cdot 7Al_2O_3$ (= $C_{12}A_7$), $CaO \cdot SiO_2$ (= CS) and $2CaO \cdot SiO_2$ (= C_2S)*) in solid reaction. The content of free lime (CaO) is therefore low at temperatures below 800° C (less than 2% by weight), rising to around 20% at higher temperatures.
The thermal dissociation of $MgCO_3$, which is of much less importance in cement manufacture, is similar to that of $CaCO_3$, but takes place at lower temperatures.

4 Solid reactions (reactions below clinkering)

From temperatures of about 550° – 600° C onwards there occur solid reactions, as already mentioned, in which the decomposition products of $CaCO_3$ react with those of the clays, at first resulting in the formation of compounds with lower content of lime (e. g., monocalcium aluminate CA, dicalcium silicate C_2S). The formation of tricalcium aluminate ($3CaO \cdot Al_2O_3 = C_3A$) and calcium aluminoferrite [$2CaO(Al_2O_3, Fe_2O_3) = C_2AF$], which occur also in portland cement clinker, begins at around 800° C. Examples of such reactions are:

$CaO \cdot Al_2O_3 + 2CaO \rightarrow 3CaO \cdot Al_2O_3$ $\quad C_3A$
$CaO \cdot Al_2O_3 + 3CaO + Fe_2O_3 \rightarrow 4CaO \cdot Al_2O_3 \cdot Fe_2O_3$ $\quad C_4AF$
$CaO \cdot SiO_2 + CaO \rightarrow 2CaO \cdot SiO_2$. $\quad C_2S$

The solid reactions proceed very slowly, but can be speeded up by: reduction of the particle size of the materials involved (i. e., larger surface area), raising of the burning temperature, presence of crystal lattice distortions.

5 Reactions in the presence of liquid phase (clinkering)

The first formation of liquid (melt), marking the start of what is known as "sintering" or "clinkering", occurs at a temperature of between about 1260° and 1310° C. With further rise in temperature the proportion of liquid phase increases to around 20 – 30% (by weight) at 1450° C, the actual proportion being dependent on the chemical composition of the material. (Thus, the proportion of liquid formed is less according as the silica modulus is higher: see Fig. 6.) At these temperatures the main component of portland cement clinker is formed, namely, tricalcium silicate (C_3S), known also as alite.
At the start of clinkering the material still contains substantial amounts of uncombined CaO as well as dicalcium silicate (C_2S). In the presence of the liquid phase these compounds pass into solution; the diffusion of the reactants is greatly facilitated in the liquid (as opposed to the solid state), tricalcium silicate (C_3S) is formed in accordance with the following reaction and crystallizes:

$CaO + 2CaO \cdot SiO_2 \rightarrow 3CaO \cdot SiO_2$ (= C_3S).

*) In cement chemistry the following abbreviated notation is employed to indicate the compounds
C = CaO, S = SiO_2, A = Al_2O_3, F = Fe_2O_3, M = MgO, Cs = $CaSO_4$, H = H_2O, N = Na_2O, K = K_2O

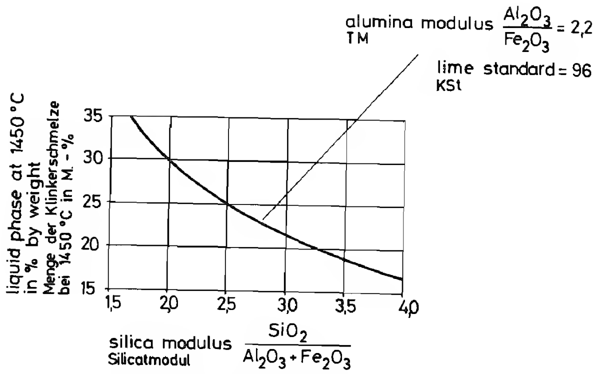


Fig. 6: Relation between the silica modulus and the content of clinker liquid phase, calculated according to L. A. Dahl, at a clinkering temperature of 1450° C (from Locher, 1979)

With this the main object of the clinkering process, i. e., the formation of the valuable compound C_3S, has been achieved, and it is this that requires and justifies the effort and cost of heating the raw materials to the high clinkering temperature. In addition, the liquid phase promotes other reactions, e. g., involving relatively coarse quartz or limestone particles.
Tricalcium silicate (C_3S) and dicalcium silicate (C_2S) are present as solid phases in the sintering liquid. At temperatures above 1400° C the liquid phase contains all the Al_2O_3 and Fe_2O_3 of the subsequent clinker and has approximately the following composition: 56% CaO, 7% SiO_2, 23% Al_2O_3 and 14% Fe_2O_3 (percentages by weight). A state of equilibrium is established at clinkering temperature.
The viscosity of the liquid phase is lower with decreasing iron modulus (alumina ratio), i. e., with increasing Fe_2O_3 content. Subsidiary mix components also affect the viscosity, which is, for example, increased by alkalis, but decreased by SO_3 and MgO. These reactions can be accelerated more particularly by:

increasing the proportion of liquid phase;
lowering the viscosity of the liquid phase;
reducing the proportion of coarse particles (especially quartz) in the raw meal.

6 Reactions during cooling

If the clinker formed in the burning process were cooled very slowly, some of the reactions already accomplished would be reversed, resulting more particularly in

the loss of tricalcium silicate – which is important to the strength development of the cement – by dissolving in the liquid. With rapid cooling, which is desirable, the liquid solidifies quickly and there is no appreciable loss of tricalcium silicate. The equilibrium is "frozen", as it were. Thus, the composition of cooled technically produced portland cement clinker is substantially similar to that attained at clinkering temperature.
In contrast with liquid phases with a high SiO_2 content, the lime-rich alumino-ferritic liquid in portland cement clinker undergoes complete crystallization even when cooled rapidly.
The rate of cooling also affects the state of crystallization, the reactivity of the clinker phases and the texture of the clinker itself. For instance, rapid cooling will produce fine closely-intergrown tricalcium aluminate (C_3A) and calcium alumino-ferrite [$C_2(A,F)$] crystals, which react slowly with water.

Other effects of rapid cooling are:

better grindability of the clinker due to stress cracks;
higher alite content because less alite is lost by dissolving;
slower setting of the cement because of intergrown finely crystalline aluminate and ferrite phases;
better soundness (less expansion) if the MgO content is above 2.5%, because more MgO is present in solid solution in the clinker, while free MgO occurs in finely crystalline form.

On the other hand, extremely rapid cooling over the entire temperature range from clinkering to ambient temperature (quenching) is liable to result in lower cement strength. It has been observed, however, that limited quenching may produce an increase in strength. The rate of cooling in the upper temperature range appears to be the important factor. In this range, relatively slow cooling under oxidizing conditions from clinkering temperature to around 1400° C (for high-alkali clinker) or around 1300° C (for low-alkali clinker) – in the kiln – is reported to have a beneficial effect on the strength of the cement, which may be attributable to crystal lattice dislocations caused by incipient decomposition of alite. (The validity of these observations and interpretations has been disputed by some authors, however.)
The rate of cooling of the clinker after leaving the kiln is generally considered not to be of appreciable influence on the strength of the cement, i. e., it does not matter which type of cooler – planetary or grate cooler, for example – is used.

7 Factors affecting the burning process

The above-mentioned reactions are affected by numerous chemical, mineralogical and physical factors, some of which can be controlled.
The **chemical composition** of the feed material supplied to the kiln has a marked influence on the burning time required. This can be defined as the length of time needed, at a certain burning temperature, to burn a raw meal of given fineness to such an extent that not more than 2% of free CaO (by weight) is present. The burning time becomes longer with increasing lime standard, silica modulus and iron modulus (the influence of the last-mentioned modulus is only slight,

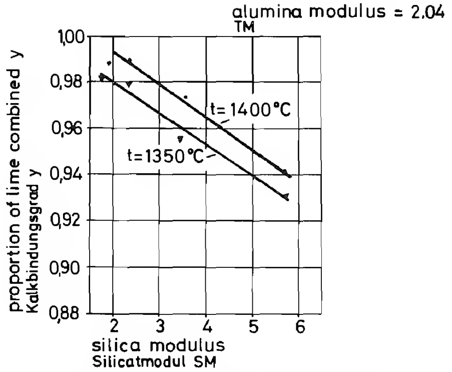


Fig. 7: Relation between the silica modulus and the combining of lime in synthetic raw meals made from pure oxides (from Syčev, 1962)

however). The relationship between the silica modulus and the combining of lime is exemplified in Fig. 7. The values represented in this diagram were obtained on synthetic raw meals in the laboratory and are only tentatively applicable to conditions in industrial cement manufacture. Alkali oxides (when present in an amount of above about 0.5% by weight) tend to inhibit the combining of lime, whereas MgO (below about 2.0% by weight) and SO_3 (below about 1.0% by weight) accelerate it in the burning process.
The **mineralogical composition**, for example, affects the pelletizability of the raw meal and also affects the water content needed in raw slurry, while the burning behaviour and the specific heat requirement are modified, inter alia, by the mineral components of the raw meal. The mineral character of clays and coarsely crystalline quartz, in particular, is a major influencing factor, but crystal lattice dislocations, crystal size and intergrowth, admixtures and impurities, natural blending of the phases in the raw material, and other factors, also play a part.
The rates at which reactions take place are generally dependent on the **particle size** of the reactants, i. e., on the reactive surface areas. Hence the raw meal should be of such fineness that in the burning process even its coarsest particles will react as completely as possible. As a rule, this condition is satisfied by cement raw meal with a residue of not more than 5–20% (by weight) retained on the 90 micron sieve, the actual maximum acceptable percentage being dependent on the composition of the meal and the type of kiln system.
Fig. 8 shows the effect of the limestone particle size on the content of free CaO at various temperatures, bearing in mind that these are values obtained in the laboratory and give only a tentative indication of conditions in actual industrial practice.

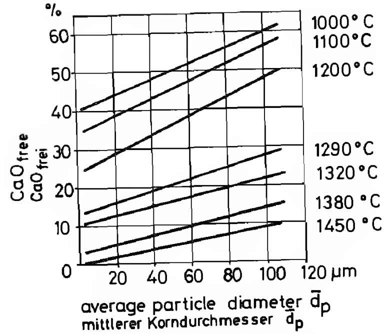


Fig. 8: Effect of limestone particle size on free CaO content at various burning temperatures (from Lehman/Locher/Thormann, 1964): dp = average particle size of a fraction; lime standard KSt I = 96; silica modulus = 3.0; alumina modulus = 2.2; $\Delta t/\Delta\tau$ = 5 to K/min.; τ = 30 min.; clay component: illite

The **homogeneity** of the raw meal is a major requirement for obtaining a clinker of uniform composition and for ensuring steady burning conditions. For this it must more particularly be ensured that the meal is of unvarying composition throughout, even within small volumetric quantities ($< 1\ mm^3$). If this is not the case, "pockets" consisting of different phases will occur in the clinker. These may consist, for example, of concentrations of free lime (which are liable to give rise to expansion phenomena on hydration) or of dicalcium silicate (belite) which in a homogeneous material would have combined to produce the desirable tricalcium silicate (alite).
So-called mineralizers as additives (e. g., fluorite CaF_2) may favourably affect the burning process.
To sum up, the burning behaviour of a raw meal is dependent on the following factors:

chemical composition (lime standard, silica modulus, iron modulus, subsidiary constituents, liquid phase, mineralizers);
mineral composition;
particle size distribution, especially the maximum particle size;
homogeneity of the raw meal;
burning conditions (rate of heating, more particularly at temperatures above 1100° C, maximum burning temperature, and retention time at this highest temperature).

The result of the burning process is portland cement clinker, consisting of the clinker phases.

References

4, 7, 8, 9, 12, 20, 23, 24, 28, 31, 33, 36, 41, 46, 49, 51, 53, 54, 59, 69, 82, 83, 87, 89, 92

IV Portland cement clinker

Portland cement clinker consists substantially of the four crystalline clinker phases alite, belite, calcium aluminate and calcium aluminoferrite in close interpenetrating association. In addition, the clinker contains voids ("pores") and usually some free (uncombined) lime; more rarely, periclase is present.

1 Clinker phases

Some important data relating to the clinker phases are given in Table 3. Fig. 9 shows the strength development of these phases. As already stated, free CaO and free MgO (periclase) may also occur in the clinker.

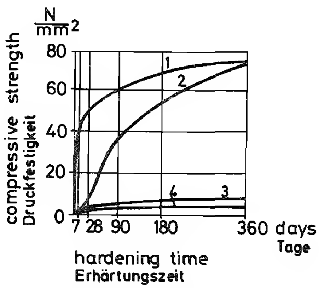


Fig. 9: Compressive strengths of clinker phases (water-cement ratio = 0.5); 1 = C_3S; 2 = C_2S; 3 = C_3A; 4 = C_4AF (from Bogue, 1955)

1.1 Alite (tricalcium silicate)

Chemically pure tricalcium silicate (C_3S)*) does not occur in portland cement clinker; it always incorporates foreign oxides, e. g., approximately 2% MgO, also Al_2O_3, Fe_2O_3, TiO_2 and others. The amounts in which these oxides are present depend more particularly on the composition of the clinker, the temperature at which it was burned and the manner in which it was subsequently cooled. They modify the properties of the alite: for example, the incorporation of foreign ions usually increases its strength. Below 1250° C, tricalcium silicate may decompose into CaO and C_2S if subjected to very slow cooling, especially if it contains Fe^{2+} as a result of burning under reducing conditions. Quantitatively and also with regard to the properties of the cement (more particularly its strength development) tricalcium silicate is the most important constituent of cement. For this compound to form in the burning process, it is essential that sintering should occur.

*) For abbreviated notation see footnote on page 123

1.2 Belite (dicalcium silicate)

Chemically pure dicalcium silicate (C_2S)*) is not found in cement clinker either; it likewise contains incorporated foreign oxides. It occurs mainly in solid form at the clinkering temperature and is present only in small proportions in clinker with a high lime standard. Its strength development is slow, but in the long run it attains strengths at least as good as those of alite. The β modification of belite, which is the form in which this compound is predominantly present in clinker, may at room temperature change into the γ modification, which is the more stable form, but virtually lacking in hydraulic properties (beta-gamma inversion). This change is accompanied by a volume increase of about 10%, which is considered to be the cause of the so-called "falling" of clinker, a rapid disintegration. This inversion can be obviated, however, i. e., the belite can be stabilized, by the incorporation of foreign ions and also by rapid cooling. With present-day technology of cement manufacture the risk of clinker falling has been eliminated.
The finely crystalline aluminate and ferrite phases are often ranked as "interstitial matter" or "matrix". Both these phases are formed from the clinker melt on cooling.

1.3 Aluminate phase

The aluminate phase (in its pure form: C_3A) likewise contains foreign ions. Here the incorporation of alkalis (Na_2O, K_2O), each in amounts exceeding 5% by weight, is possible. The aluminate phase possesses a high degree of reactivity, which is further increased by the incorporation of alkalis. The presence of the phases NC_8A_3 and KC_8A_3 has been reported. In order to retard the reaction of the aluminate phase at the start of hydration, every cement must contain some added sulphate (e. g., in the form of gypsum) as a setting retardant.
Together with alite and belite, the aluminate phase may somewhat increase the early strength of the hardening cement (this effect being due to the considerable heat of hydration that this compound evolves). Its own hydraulic properties are slight, however.
The compound $C_{12}A_7$ may also occur.

*) For abbreviated notation see footnote on page 123

Table 3: Clinker phases

designation of the phase in the clinker of the pure phase	alite tricalcium silicate	belite dicalcium silicate	aluminate phase tricalcium aluminate	ferrite phase calcium aluminoferrite
composition of the pure phase	$3\ CaO \cdot SiO_2$	$2\ CaO \cdot SiO_2$	$3\ CaO \cdot Al_2O_3$	$2\ CaO(Al_2O_3\ \ Fe_2O_3)$
abbreviated notation	C_3S	C_2S	C_3A	$C_2(A,F)$ or $C_2A_pF_{1-p}$
foreign ions commonly incprporated in clinker phases	Mg, Al, Fe	alkalis, Al, Fe, fluoride	alkalis, Fe, Mg	Si, Mg
number of modifications	6	5	3	1
modifications occurring in technical clinkers	monoclinic (M II) trigonal (R)	β-belite, monoclinic (α and α' belite)	cubic orthorhombic tetragonal	orthorhombic
colour of the pure phase	white	white	white	dark brown due to MgO incorporation: dark grey-green
proportions in portland cement clinker (% by mass)				
maximum	80	30	15	15
average	60	15	11	8
minimum	40	0	7	4
technical properties in cement	rapid hydration, high initial and good final strength, moderate heat of hydration, main strength constituent in normal portland cement	slow hydration, good final strength, low heat of hydration	rapid hydration, high heat of hydration which promotes early strength, shrinks appreciably on hydration, reacts with sulphates and thus undergoes volume (expansion)	slow and moderate hydration, hardly any strength development, moderate heat of hydration, ion, gives normal cement its colour

1.4 Ferrite phase

The ferrite phase does not possess a constant chemical composition; it is in fact a member of a solid solution series extending theoretically from C_2A to C_2F (C_2A is still not existing):

$C_2A \ldots C_6A_2F \ldots C_4AF \ldots C_6AF_2 \ldots C_2F$.

Depending on the availability of iron and aluminium, the members of the solid solution series will be situated nearer the iron-rich or nearer the aluminium-rich end thereof. Quite often the composition of this phase in cement clinker corresponds more or less to C_4AF. The general formula of the series is $C_2(A,F)$ or $C_2A_pF_{1-p}$. Foreign ions are incorporated in the ferrite phase as well. It is the phase that contributes more particularly to giving cement its colour: pure $C_2(A,F)$ is brown, $C_2(A,F)$ containing MgO is of a dark grey/green colour. It is very slow-reacting and of little importance to the properties of the cement.

1.5 Other clinker phases

Most cement clinkers contain **free CaO (uncombined lime)** in amounts up to 2% by weight. Its presence is due either to unsuitable preparation of the raw meal (inhomogeneous or too coarse), to inadequate burning (so that it was not combined by other oxides), to too slow a rate of cooling (so that partial decomposition of C_3S or C_3A could occur) or to too high a lime content (LSt III > 100). Free lime is undesirable in appreciable concentrations (above about 2.5% by weight), as it is liable to cause expansion phenomena in mortar and concrete (lime expansion), $[CaO + H_2O \rightarrow Ca(OH)_2]$.

MgO-rich clinkers may contain **free MgO (periclase)**. Since about 2.0 to 2.5% MgO by weight is combined in the form of a solid solution in the other phases of the clinker, a cement conforming to the standard specifications may permissibly contain up to about 2.5–3.0% of peroclase (according to German Standard DIN 1164, up to a total of 5.0% MgO by weight is allowed). The proportion of MgO that is combined in other phases will depend on the chemical composition of the clinker and its conditions of production. Periclase is undesirable because, if present in substantial amounts, it may cause expansion similar to that caused by lime (magnesia expansion), but more surreptitious because in some cases the damage it causes may remain undetected for years.

Finely crystalline and uniformly distributed periclase causes less expansion than does an equal quantity of periclase that is present in coarsely crystalline form or in local accumulations ("pockets"). The same is true of free lime and its expansion effects.

The expansion due to free CaO is a result of its hydration, similar in principle to slaking, but slower: it reacts with water to form $Ca(OH)_2$, which has about twice the volume of the CaO from which it was formed. Magnesia expansion is similarly due to the reaction of MgO with water. The expansion effects are commonly referred to as "unsoundness" of the cement.

In rare cases cement clinker may moreover contain small amounts of, for example, alkali sulphates and glassy phase.

By way of example, Table 4 gives the chemical compositions of the phases of a portland cement clinker.

Table 4: Experimentally determined chemical composition of the clinker phases of a portland cement clinker (% by weight)

	alite	belite	aluminate phase	ferrite phase
CaO	69.70	63.20	59.50	51.40
SiO_2	24.90	31.50	4.21	2.28
Al_2O_3	1.12	1.84	27.52	19.60
Fe_2O_3	0.64	0.96	5.76	22.52
MgO	0.89	0.48	0.85	3.18
K_2O	0.19	0.75	0.66	–
Na_2O	0.06	0.19	0.25	–
TiO_2	0.16	0.24	0.48	1.60
P_2O_5	–	0.28	–	–

2 Judging the quality of clinker

Various methods of judging the quality of cement clinker are available. As a rule, several are applied.

Complete chemical analysis (by wet-chemical analysis or X-ray fluorescence analysis) gives information on the overall composition. From the results it is possible to calculate the **lime standard** and the moduli (silica modulus, iron modulus) which together provide more conveniently assimilable information on the quality of the clinker (see also Section II.2.1). The **potential phase composition**, as envisaged by Bogue, can also be calculated from the analytical results. This calculation presupposes that the clinker melt (liquid phase in clinkering) crystallizes in equilibrium with the solid phases and that the clinker phases are of chemically pure and stoichiometric composition, i. e., pure C_3S, C_2S, C_3A and C_4AF. In reality the first assumption (equilibrium on crystallization) is not fulfilled, as was pointed out in Section III.6 dealing with the reactions on cooling; nor is the requirement of chemical purity, for the clinker phases contain incorporated foreign ions. All the same, this phase calculation yields reasonably useful approximate values for guidance. As a rule, the actual alite content is higher, the belite content lower than calculated, whereas the actual content of the aluminate and ferrite phases differs only by a few per cent from the calculated ("potential") content (see Table 5).

Table 5: Comparison of potential and microscopically determined (actual) phase compositions of various portland cement clinkers (% by weight)

phase	normal portland cement clinker		MgO-rich portland cement clinker		K_2O-rich portland cement clinker	
	pot.	micr.	pot.	micr.	pot.	micr.
alite	49	70	42	58	39	51
belite	21	7	26	21	29	19
aluminate phase	13	11	15	12	17	22
ferrite phase	11	10	11	9	13	8

Bogue's formulas for calculating the potential composition:

For normal portland cement clinker:

$C_3S = 4.071\ CaO - 7.602\ SiO_2 - 6.719\ Al_2O_3 - 1.430\ Fe_2O_3$
$C_2S = 8.602\ SiO_2 + 5.068\ Al_2O_3 + 1.079\ Fe_2O_3 - 3.07\ CaO$
or $C_2S = 2.868\ SiO_2 - 0.754\ C_3S$
$C_3A = 2.650\ Al_2O_3 - 1.692\ Fe_2O_3$
$C_4AF = 3.043\ Fe_2O_3$.

For clinker with iron modulus 0.64 (rich in iron oxide, no C_3A)

$C_3S = 4.071\ CaO - 7.602\ SiO_2 - 4.475\ Al_2O_3 - 2.863\ Fe_2O_3$
$C_2S = 2.867\ SiO_2 - 0.754\ C_3S$
$C_2F = 1.702\ Fe_2O_3 - 2.665\ Al_2O_3$
$C_4AF = 4.766\ Fe_2O_3$.

For the oxide symbols in these formulas the respective analytical results (in % by weight) should be substituted. If the content of free lime is known, this should be subtracted from the overall CaO content before the calculation is done.
If negative values are found for C_2S, it means that free lime must be present.
Since the alkalis, MgO and other subsidiary constituents are not taken into account in the calculation, the potential phase content is always found to be below 100%.

Example of the calculation of the potential phase composition. Consider a normal portland cement clinker with the following chemical analysis (% by weight):

loss on ignition	=	0.42	MgO	=	2.00
insoluble in HCl	=	0.15	K_2O	=	0.95
SiO_2	=	20.90	Na_2O	=	0.21
Al_2O_3	=	6.05	SO_3	=	0.54
Fe_2O_3	=	3.20	CaO_{free}	=	1.05
CaO	=	64.55	residue	=	1.03

Calculation:

$C_3S = 4.071 \times 63.50 - 7.602 \times 20.90 - 6.719 \times 6.05 - 1.430 \times 3.20 = 54.4\%$ (say 54%)
$C_2S = 2.868 \times 20.90 - 0.754 \times 54.4 = 18.9\%$ (say 19%)
$C_3A = 2.650 \times 6.05 - 1.692 \times 3.20 = 10.6\%$ (say 11%)
$C_4AF = 3.043 \times 3{,}20 = 9.7\%$ (say 10%)

Sum of the clinker phases = 93.6% (say 94%)
(percentages by weight).

Another important criterion is the **free lime content** (uncombined CaO), which is determined by wet-chemical analysis or X-ray diffractometry. In conjunction with the lime standard it gives information on the production conditions, more particularly the degree of burning. The free lime content is not allowed to exceed a certain limiting value which is in the range of 2 to 3% (by weight), depending on

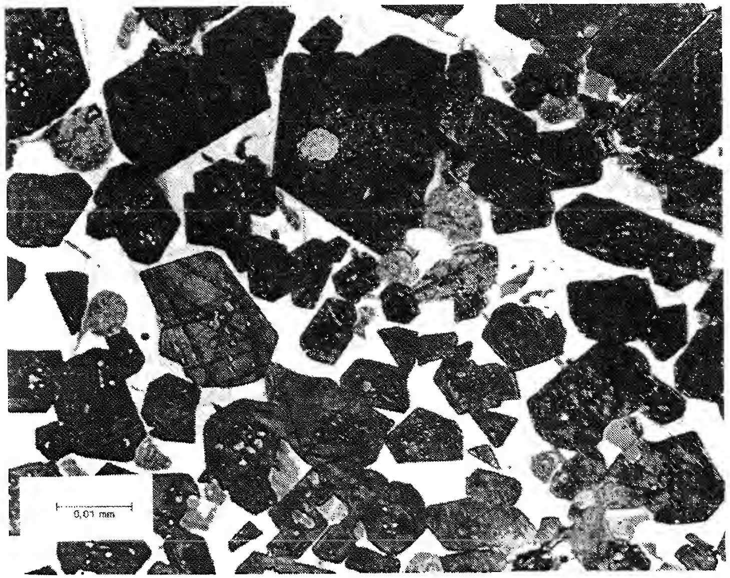


Fig. 10: Portland cement clinker: micrograph obtained with reflected light: alite: dark grey, mostly with straight boundaries; belite: light grey, curved boundaries; ferrite: white matrix; aluminate: dark inclusions in white matrix

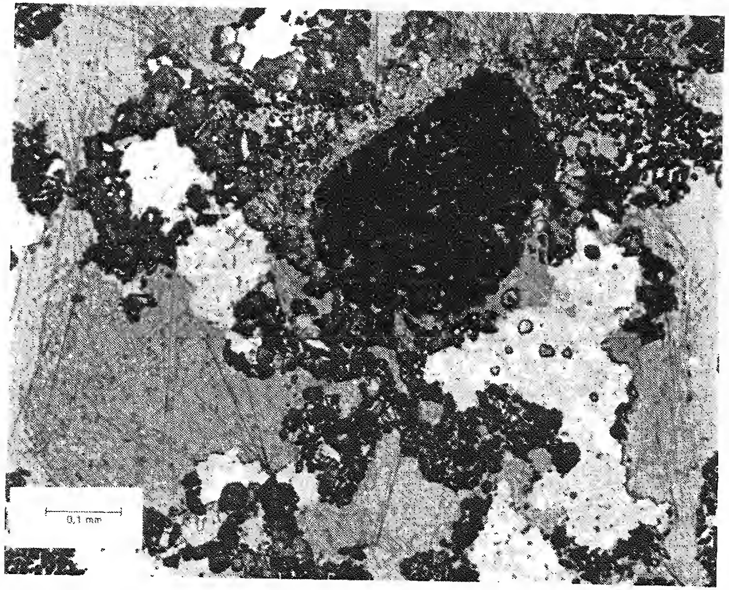


Fig. 11: Portland cement clinker: under-burned (porous); micrograph obtained with reflected light: free lime: black pocket; belite: light-coloured textured areas; alite: dark textured areas; pores (here filled with resin): grey areas with grinding scratches

the production conditions, for otherwise the risk of lime expansion in the mortar or concrete made with the cement cannot be ruled out. The factors causing the presence of free lime are explained in Section IV.1. The test for lime expansion is described in Section X.3.

The **bulk density** of a particular particle size fraction of clinker (e.g., 5–7 mm), obtained by screening, provides a check on the degree of burning. Depending on the raw meal (chemical composition) and characteristics of the kiln plant (porosity of the clinker, etc.), the values for the bulk density of adequately burned clinker range between 1.2 and 1.6 kg/dm^3. The permissible minimum value in any given case has to be determined empirically.

Microscopic examination of the clinker yields information on the nature, conformation and distribution of the clinker phases. The quantitative proportions of these phases depend on the chemical composition of the clinker, whereas their conformation and distribution are determined by the production conditions

(fineness, particle size distribution, maximum particle size and homogeneity of the raw meal, heating-up rate, duration of sintering, cooling rate, etc.). Experts can detect certain defects in the production conditions by microscopic examination of the clinker and decide on ways and means of overcoming them. As a rule, polished and etched specimens are employed, which are examined by reflected light at magnifications of between 50 and 1000. Properties such as shape, reflectivity, hardness, etching behaviour (e.g., in water or in a solution of nitric acid in alcohol), etc. are used as means of identifying the phases and also yield other information on them. Figs. 10 and 11 are micrographs of portland cement clinker in reflected light.

The constituent clinker phases can be determined qualitatively, and also to a great extent quantitatively, by means of **X-ray diffractometry** (diffraction analysis). The quantitative determination of free lime for production control purposes by this method has acquired practical importance. To use this method for quantitatively determining all the clinker phases presents difficulties, because various important diffraction lines coincide (e.g., those of alite and belite), because the incorporation of foreign ions causes variations and because the degree of disorder in the structure of the various clinker phases differs in consequence of varying production conditions.

References

3, 8, 19, 23, 25, 28, 30, 31, 33, 34, 36, 39, 41, 42, 45, 46, 53, 57, 67, 69, 83, 84, 87, 92, 93

V. Finish grinding

1 The materials involved in finish grinding

1.1 Portland cement clinker

With the exception of high-alumina, all standard cements contain portland cement clinker. This material has been dealt with in Section V.

1.2 Blastfurnace slag

Blastfurnace slag, more particularly in granulated form, is a so-called latently hydraulic material, i.e., it needs an activator to enable it to harden "hydraulically". In practice, calcium hydroxide (in cement clinker or as hydrated lime) and sulphates (gypsum, anhydrite) are used as activators. Slowly cooled crystalline blastfurnace slag in lump form is unsuitable, however; to possess latent hydraulicity, the slag has to be in a glassy form produced by rapid cooling. This is achieved by quenching the molten slag in water, which yields a granulated

product. The granulated blastfurnace slag should have the lowest possible residual water content (favourable values are below 10%). The particle size is usually below 3 mm.
The hydraulic properties of blastfurnace slag are determined by its chemical composition and its glass content. The latter should be above 90%. Methods of producing slag with 95–100% glass content are now available.
The chemical composition of the granulated blastfurnace slags used in cement manufacture is approximately in the range indicated in Table 6. There are formulas for estimating the hydraulic properties on the basis of the chemical analysis of the slag. According to DIN 1164 a granulated slag is to be classed as suitable for making slag cements (more particularly the two German varieties known as "Eisenportland" cement and "Hochofen" cement) if the following condition is satisfied:

$$\frac{CaO + MgO + Al_2O_3}{SiO_2} \geq 1 .$$

However, such formulas can do no more than give approximate guidance. So far, it has not proved possible to establish a generally-valid formula that will reliably predict the hydraulic properties on the basis of the chemical analysis data, nor does it appear likely that such a formula will be found. In general terms, however, it can be said that the hydraulic properties are better according as the content of CaO, MgO and Al_2O_3 is higher (this applies for MgO only up to about 12%, while Al_2O_3 above 13% improves only the early strength).

Table 6: Chemical compositions of the granulated blastfurnace slags used in cement manufacture (% by weight)

oxide	content	oxide	content
SiO_2	28–38	CaO	35–48
Al_2O_3	9–18	MgO	2–10
FeO	0– 2	S	1– 3
MnO	0– 2	Na_2O	0– 2

A more reliable method of determining the hydraulic properties of a granulated blastfurnace slag consists in intergrinding it with clinker and gypsum to produce a slag cement with a high slag content (in the laboratory) and testing this cement for strength and, if necessary, for other properties as well. For comparison, a "cement" may be made which contains, instead of slag, an equal quantity of an inert substance (e.g., quartz sand) of the same fineness or, alternatively, a portland cement made from the same clinker, but without slag, may be ground to the same fineness as the slag cement and tested.

1.3 Pozzolanas

Pozzolanas are materials, mainly of natural origin, which react at normal temperature with calcium hydroxide and thus produce strength-developing chemical compounds (hydraulic hardening). Most pozzolanas are volcanic materials, especially those known as tuffs. The name "pozzolana" is derived from Pozzuoli near mount Vesuvious on the Gulf of Naples. In Germany, similar materials known as Rhenish trass (a volcanic tuff from the Neuwied Basin near Koblenz) and Bavarian trass (a rock transformed by meteorite impact, found in the area called Nördlinger Ries, about 80 km south of Nuremberg) are used as additives to cement. **Trass** has to conform to German Standard DIN 51043. Burned oil shale residue, used more particularly at Dotternhausen near Donaueschingen, is another pozzolanic material that calls for mention. In other countries such materials comprise, besides volcanic rocks, various siliceous sedimentary deposits, including more particularly kieselguhr (diatomaceous earth consisting of the remains of unicellular creatures with siliceous skeletons). Essential quality requirements of a pozzolana are that it contains large amounts of SiO_2 and Al_2O_3 in a suitably reactive form, so that it can react with $Ca(OH)_2$. The suitability of such materials as ingredients of cement can be determined by means of comparison tests (as with blastfurnace slag) or by chemical methods (testing the capacity to combine with lime).

1.4 Fly-ash

Fly-ash or pulverized fuel ash (PFA) is obtained, for example, in dust collection equipment of furnaces fired with pulverized coal, especially those of electricity generating plants. It is composed of glass-like particles of predominantly spherical shape and consisting mainly of SiO_2, Al_2O_3 and Fe_2O_3. It is a pozzolanic material which is activated by calcium hydroxide and is then capable of hydraulic hardening. This applies more particularly to the glass content of the ash, which should therefore be as high as possible. On the other hand, it should contain the least possible amount of burnt carbon residue, as this is detrimental to the cement properties (lower strength and durability of concrete made with the cement).
The reactivity of fly-ash is higher according as its specific surface is larger. For most types of fly-ash this is between about 1000 and upwards of 4000 cm^2/g (Blaine), though it should be noted that these values may be falsified or shifted to higher values by the presence of carbon particles. The ash particle sizes are generally between 0.5 and 200 microns. Coarse-graded fly-ash can be improved by grinding, preferably by intergrinding with portland cement clinker and gypsum to produce the desired cement. Up to about 30% of fly-ash – depending on the quality and properties of the ash – may thus be incorporated as an additive in cement.

1.5 Sulphates

A quantity of sulphate (in the form of gypsum or a mixture of gypsum and anhydrite-II) is always added to the portland cement clinker in the finish grinding

process, the object of this addition being to control (retard) the setting time of the product. The retarding effect is brought about by a reaction of the sulphate with the tricalcium aluminate, which would otherwise set too quickly (clinker containing a higher content of C_3A will require more sulphate; see also Section VII.2). However, too much sulphate in the cement is liable to cause expansion phenomena (Section VII.2), and for this reason upper limiting values are specified for the cement content (reckoned as SO_3). The values laid down in DIN 1164 are given in Table 7. Natural impurities in raw gypsum (e.g., clay, calcite) do not adversely affect the quality of the cement. Depending on the C_3A content, the fineness of the cement and the alkali content, there exists for every cement a certain optimum sulphate content which may moreover distinctly improve the strength. This optimum content of sulphate is higher according as the C_3A and alkali content of the clinker is higher and the cement is more finely ground. Because of the differences in solubility between hemihydrate (highly), gypsum (moderately) and anhydrite-II (poorly soluble), the nature of the sulphate-bearing compound added to the clinker is also of some importance. The optimum sulphate content will be higher if anhydrite-II is used. In order to avoid possible irregularities of setting, it is preferable to use mixtures of gypsum and anhydrite-II (in proportions ranging from about 1:1 to 1:8). For cement with a high content of C_3A and alkalis and ground to a high degree of fineness the optimum sulphate content is around 5% SO_3 by weight. For coarsely ground cement containing little or no C_3A and with a low alkali content the SO_3 requirement is in the region of 2.5–3% by weight.

Table 7: Highest permissible SO_3 content in cements (DIN 1164)

type of cement	highest permissible SO_3 content in % by weight for specific surface[3]) of the cements	
	from 2000 to 4000 cm^2/g	over 4000 cm^2/g
portland cement, Eisen portland cement, trass cement	3.5	4.0
Hochofen cement with 36 to 70% by weight of blastfurnace slag	4.0	
Hochofen cement with more than 70% by weight of blastfurnace slag	4.5	

2 Fineness and particle size distribution

Under otherwise similar conditions a substance will react more rapidly in proportion as its specific surface (in cm^2/g) is larger. For this reason the raw materials for cement manufacture have to be ground before burning, and the clinker (with admixtures, especially gypsum) has to be ground to suitable fineness in order to produce a cement that will react readily with water in the hydration process. Thus, one and the same clinker will achieve better (more rapid) strength development according as it is more finely ground, i.e., acquires a larger specific surface. For every additional 100 cm^2/g of specific surface the gain in strength of the cement is in the region of 0.5 to 2.0 N/mm^2, the average increase in 28-day compressive strength being approximately 1 N/mm^2. The same applies to all the usual standard testing ages for cement. Only after a much longer period (several years), when even the coarser particles have fully reacted, is there likely to be little difference in the strength finally attained by coarser and finer cements. Reference values for cement fineness are given in Table 8.

Table 8: Reference values for fineness of cements

cement	percentage (by weight) retained on 0.09 mm standard sieve (DIN 4188)	specific surface (Blaine) in cm^2/g
portland cement 35	< 10	2400–4000
Hochofen cement 35	< 6	3000–4000
portland cement 45	< 6	2800–4500
Hochofen cement 45	< 3	3300–4500
portland cement 55	< 1	4000–6000
Trass cement	< 4	3000–5500

In clinker grinding, the gypsum, being more readily grindable, tends to be concentrated in the finer particle size fractions of the product. So does any fly-ash that may be added, whereas blastfurnace slag becomes concentrated in the coarser fractions.

Strength development, especially the early strength, is distinctly improved if the cement is more closely graded, i.e., if the middle range of particle sizes between 3 and 30 microns is increased to above 50%, say, at the expense of the coarser and the finer particles – provided that the specific surface of the cement is not reduced. The improvement is due to the faster rate of hydration achieved. For producing such closely graded cement it is essential that the grinding plant has a highly selective classifier (air separator). The use of grinding aids is reported also to be helpful in achieving this result. However, the effect of grading (particle size distribution) on the strength development of industrial cements is not always clearly manifest. Fig. 12 shows the strength development of various granulometric classes of cement.

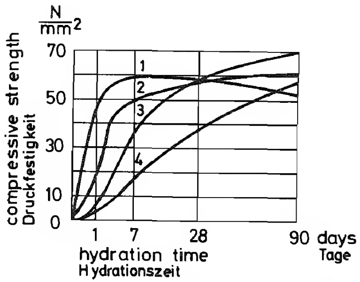


Fig. 12: Strength development during the hydration of cements of various granulometric classes (from Sweden); 1 = 0/3μm, 2 = 3/9μm, 3 = 9/25μm, 4 = 25/50μm)

Clinker which has been stored under damp conditions for some considerable time already contains hydration products. When such clinker is ground, these products tend to become concentrated in the finest fractions (and cause high specific surface values) while furthermore, e.g., by forming coatings on the grinding media, they obstruct the grinding of the unhydrated clinker constituents which thus tend to form higher concentrations in the coarser fractions. For these reasons the specific surface values yielded by such clinker should be rated with some caution: in general, higher values should be aimed for than in the grinding of fresh unhydrated clinker.
The fineness of grinding of cements may be determined by sieving or a separation method (the fineness being expressed as a certain percentage by weight above a certain size, e.g., as residue retained on a standard sieve), but is more usually based on the specific surface determined by the Blaine method (air permeability of a bed of cement, the result being expressed in cm^2/g; the finer the cement, the higher the specific surface) (see also Section X.1).

3 Mill atmosphere

Heat is generated in the grinding of cement clinker, resulting in a rise in temperature, which may in some instances exceed 120° C. The water content of the gypsum ($CaSO_4 \cdot 2H_2O$) is driven out, slowly at first (from about 40°–45° C onwards), but above 80° C at a rapid rate, as a result of which the gypsum is partly or indeed completely dehydrated (the latter above 110° C), so that it is transformed into hemihydrate ($CaSO_4 \cdot {}^1/_2H_2O$) or anhydrite III ($CaSO_4$, soluble anhydrite), see Fig. 13. These partly or wholly dehydrated sulphates dissolve much more easily in water than gypsum does and are thus more reactive. In C_3A-rich cements this

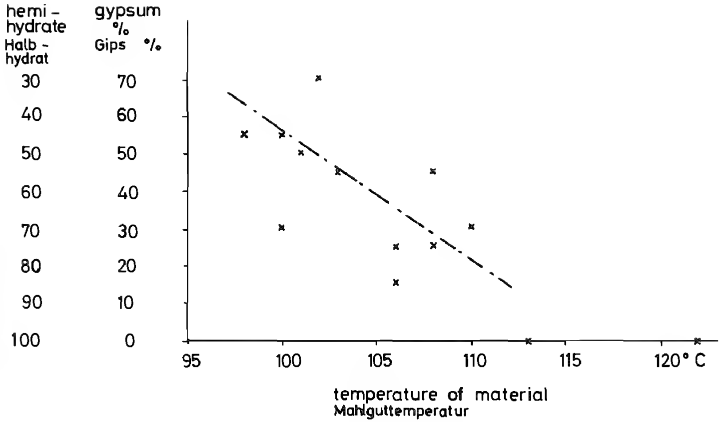


Fig. 13: Content of gypsum and hemihydrate (including soluble anhydrite) as a function of mill charge temperature during grinding

reactivity is advantageous for its retarding effect on the setting behaviour (C_3A reacts with sulphate to form ettringite: see also Section VII.2). On the other hand, when "hot"-ground cement is mixed with water, a solution supersaturated with calcium sulphate may quickly develop, from which gypsum is then precipitated in the form of needle-shaped crystals which interlock and cause a stiffening of the mass called **"false set"**. This is, however, a temporary phenomenon, which can be reversed by further mixing. Rapidly forming needle-shaped crystals of syngenite and ettringite may also have a share in this early stiffening, and the alkali sulphates contained in the clinker react just as quickly.
In portland cement of normal composition the SO_3 content corresponding to hemihydrate, anhydrite III and clinker alkali sulphate together should be below 2.2–2.5% by weight (depending on the C_3A content; a lower limit is applicable to portland blastfurnace cement, cements with low C_3A content and certain others). If this limiting value for the combined SO_3 content is exceeded, there is a risk of false set. Apart from remining below this limit, other ways to overcome this problem are: increasing the mixing time of the cement or, at the clinker grinding stage, substituting anhydrite-II for a proportion (up to about 50% by weight) of the gypsum.
The moisture given off by the gypsum dehydration to the atmosphere in the mill, as also the water which may be injected into the mill for cooling its charge during grinding, will react more particularly with the finest particles of the cement formed. As a result, the reactivity of the tricalcium aluminate (C_3A) is in part substantially reduced, and if fairly large amounts of moisture are thus released into the mill, the

strength development of the cement will be appreciably affected (strength losses of more than 10% are liable to occur). C_3A-rich and alkali-rich cements are especially prone to this effect.

4 Grinding aids

Grinding aids have been used in Germany for the last twenty years or so, more particularly for the grinding of cements in the higher ranges of fineness (specific surface above about 3500 cm^2/g). Their usefulness is greater according as the cement to be ground is finer. For equal cement fineness, grinding aids can sometimes substantially increase mill throughput (Table 9). However, they must be suitably tested with regard to their harmlessness in concrete made with the cement; more particularly, they must not promote corrosion of reinforcing steel, and a certificate to that effect must be supplied by the manufacturers of the grinding aids. Whereas the advantageous action of these substances in connection with the grinding of portland cement is beyond dispute, they are of relatively little value in the grinding of slag cements.

Table 9: Average increase in performance of finish grinding mills as a result of grinding aids (from Schneider, 1969)

cement	specific surface cm^2/g	increase in throughput %	amount of additive %
PZ 35	2400–3000	bis 10	0,01–0,03
PZ 45	3000–4000	10–30	0,02–0,06
PZ 55	4000–5500	25–50	0,04–0,1

Particularly effective grinding aids are glycols (e.g., ethylene glycol, propylene glycol) and ethanol amines (e.g., triethanol amine). As a rule, they are added in quantities of less than 0.005% by weight. Larger additions (above 0.2% by weight) of triethanol amine are liable to lower the early strength, but the 28-day strength is not adversely affected. Grinding aids have been used for a good many years, as already mentioned, and it has been established that they do not impair the long-term behaviour of concrete either.

References

4, 8, 18, 23, 37, 28, 37, 43, 44, 46, 47, 50, 53, 55, 58, 61, 62, 72, 73, 74, 78, 83, 87.

VI. Storage of cement

1 Storage in the cement works

The finished cement that is discharged from the clinker grinding mill is stored in silos in which it should, ideally, undergo no subsequent changes. However, certain influences may act upon the cement in storage and have a detrimental effect on its quality.
Cement should be stored at the lowest possible temperature.

At temperatures of 50°–60° C there is little dehydration of gypsum for storage periods of up to about 28 days, but at 80°–90° C the dehydration is very considerable, and under such conditions the gypsum may lose all its crystal water. Indeed, incipient dehydration is found to occur in cement stored at only 40° C for periods in excess of 28 days. The water thus released – together with moisture of atmospheric and possible other origin – reacts with the cement in the cooler zones of the silo. The C_3A in the cement is more particularly prone to react with water. Acicular (needle-shaped) ettringite and syngenite are formed, also tabular aluminate hydrate. These newly formed crystals are liable to **cause solidification** of the (formation of **lumps**, "bridging" in the silo). Since the compounds which are more particularly involved in these solidification reactions with water are C_3A and alkali sulphates, cements which have a high content of these compounds are notably prone to be affected in this way.
Water absorption by cement, especially if the latter has a high C_3A content, furthermore causes retardation of setting (because of diminished reactivity of the C_3A) and, depending on how much water is absorbed, also causes loss of strength (in consequence of pre-hydration of the C_3S in the cement). Besides, **false set** – temporary early stiffening of the cement when mixed with water – may also be due to causes associated with silo storage (see Section V.3).
In general, cement is less likely to be affected by storage according as it is more finely ground. This may appear somewhat surprising, but the reason is that in finer cement the average radius of the pores or voids between the cement particles is smaller, so that water vapour diffuses less easily through the bed of cement.
As can be inferred from the foregoing, as little moisture as possible should be allowed to get into the cement storage silo, and the temperature of the stored cement should, if possible, be below 60° C. To minimize the access of water to the cement, it may be advisable to reduce the gypsum content or to substitute anhydrite II for some of the gypsum in clinker grinding, the amounts of water (if any) that are sprayed into the mill should be duly monitored, and the feed of moist clinker and/or blastfurnace slag to the mill should be avoided.
Cement which, when fresh, has normal setting properties may become quick-setting as a result of storage. This is more particularly liable to occur in the following types of cement:

(1) Cements produced from clinker whose molar ratio $(K_2O + Na_2O) : SO_3 > 1$. In this case the change from normal to quick setting behaviour may be caused by alkali carbonate (formed possibly via alkali aluminate).

(2) Cements with low C_3S and high C_3A and $C_2(A,F)$ content. After storage in air at low humidity values (relative humidity below about 50%) a diminished reactivity of the C_3S, characterized by less formation of $Ca(OH)_2$ at the start of hydration, may occur in conjunction with unimpaired intensive reactivity of the C_3A.

These causes may be superimposed, and other causes may be involved as well. The following counter-measures are available: changing to raw materials of different composition (in particular, a low alkali content) and using water-repellent admixtures in the clinker grinding mill, so that the cement is rendered "hydrophobic" and thus insensitive to moisture.

2 Storage on the construction site

Cement which is stored unprotected for any considerable length of time will absorb moisture, causing lumps to form and resulting in a loss of hardening capacity. So long as the lumps are friable – easily crumbled between the fingers – the decline in strength is not serious, however.

Cement in sacks is more at hazard than bulk cement in a bin or silo. Hence properly dry storage conditions for sacks of cement are important: under cover in a shed or, if in the open, placed on battens clear of the ground and covered with plastic sheet. Cement thus stored in sacks, or in a bin, on the construction site undergoes a loss of strength averaging somewhat over 10% in three months. The decrease in early strength, especially in the case of more finely ground cements, may be greater than this. For this reason the period of storage should always be kept as short as possible, and for very fine cements it should preferably not exceed one month or at most two months.

References

1, 2, 4, 8, 13, 21, 26, 28, 32, 63, 68, 79, 83, 87.

VII. Hydration of cement (setting, hardening, strength)

1 General

Hydration is a process in which water is combined with the reacting substance. The hydration of cement is accompanied by solidification, i. e., an initially liquid or plastic system (cement paste) progressively turns into a stone-like solid (referred to as hardened cement paste). The process of solidification comprises two stages: setting and hardening. On setting, the cement paste stiffens into a solid, but as yet of negligible strength. In the then following stage of hardening the paste gradually develops considerable strength. There is no sharp division between setting and hardening, the transition is gradual.

In the hydration and solidification of cement a number of different processes actually take place simultaneously and/or successively. These include more particularly:

– chemical reactions: especially hydration and hydrolysis reactions;
– dissolving and crystallization processes: gel-like and crystallized newly formed substances containing water (hydrate phases) are formed from supersaturated solutions and in topochemical processes;
– interfacial processes: surface attractive forces (adhesion) produce bonding of the constituents of the cement paste.

The hydration reactions are exothermic, i. e., heat is evolved. The heat evolution of cement hardening under adiabatic test conditions attains a maximum after 1 to 3 days and then proceeds at a diminishing rate. The heat given off, in terms of quantity and in relation to time, depends on the type of cement (more particularly its constituent phases), its fineness and the presence of additives, if any (blastfurnace slag, pozzolana).

The overall result of the hydration reactions is a hardened product possessing high strength. The strength of the hardened cement paste is primarily due to its internal structure, which in turn is determined by the shape and size of the hydration products (hydrate phases) and their spatial arrangement and packing density (porosity). The water that has to be added to the cement in order to achieve hydration is combined chemically as hydration water or as hydroxide. The theoretically required amount of water is not more than about 30% of the weight of the cement (water-cement ratio $w/c \leqq 0.3$). Besides this chemically combined water, however, a certain amount of water is physically bound on the very large surface areas of the hydrate phases (adsorbed water, corresponding approximately to $w/c \leqq 0.1$). Also, some water is present as capillary water in the voids of the hardened cement paste. The higher the capillary water content (leaving capillary "pores" after evaporation), the lower will be the strength, the resistance to chemical attack and the frost resistance of the hardened paste or more particularly the concrete or mortar in which it forms the bonding medium. Also, these pores increase the permeability to water. Fig. 14 shows how the strength decreases with increasing water-cement ratio.

The final strength of the hardened cement paste under normal conditions of hardening (normal temperature, not under pressure) is at best about 200 N/mm^2, as laboratory research has established. The principal influencing factor is the capillary porosity (which in turn is bound up with the water-cement ratio and with the degree and progress of hydration), while the composition of the cement and the conditions of hardening are subsidiary factors in connection with strength development. In actual practice, as distinct from the laboratory, the final strength attained is generally less than the above mentioned value.

Under practical conditions the strength of mortar (aggregate particle size < 4 mm) and concrete (aggregate particle size usually < 16 mm, < 32 mm or < 63 mm) is affected more particularly by the following factors:

type and quality of the cement;
water-cement ratio (proportions by weight),

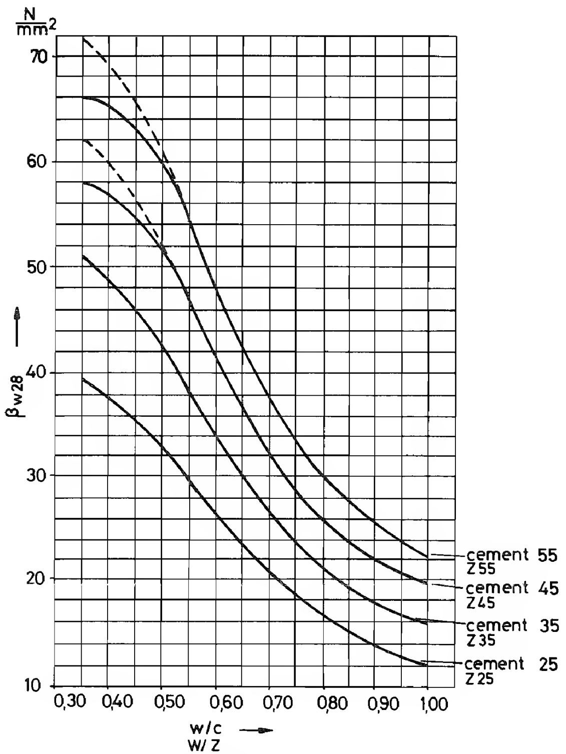


Fig. 14: Relation between 28-day compressive strength of concrete (β_{w28}), water-cement ratio and cement strength class (from Graf)

aggregates (type, strength, particle shape, surface, quantity, grading);
admixtures and additives, if any*);
compaction and curing;
temperature and age.

*) In concrete technology, "admixture" and "additive" are often treated as synonymous terms, but sometimes (as also in this translation) a distinction is drawn between substances such as plasticizers, retarders, etc. added in very small amounts ("admixtures") and substances such as trass, fly-ash, etc. which form a quantitatively more substantial component of the cement ("additives")

The hardening of cement can be accelerated or retarded by the incorporation of **admixtures** of various kinds in the mix. Hardened cement paste, and therefore the mortar or concrete in which it forms the bonding medium, is a stable substance, resistant to normal environmental conditions. Certain external influence may, however, have a harmful effect, causing **concrete corrosion.**

References

4, 6, 8, 13, 23, 28, 34, 35, 38, 40, 46, 53, 83, 90.

2 Hydration of the clinker phases

For a fuller explanation of the hydration process it will be necessary to take a look at the four principal clinker phases: alite, belite, aluminate and ferrite. In general, the hydration reactions can be represented as follows in a simplified general way:

clinker phases +	water →	hydrate phases +	energy
(high in energy; contain no water)		(low in energy; contain water)	(heat of hydration)

The progress of the reaction can be measured with reference to newly formed compounds, heat of hydration evolved, chemically combined water, strength development.
Especially important are the hydration reactions of aluminate and of alite. Belite reacts in the same manner as alite, while ferrite is of no great significance.

2.1 Aluminate

In the absence of gypsum in the cement, tricalcium aluminate reacts very quickly:

$$3\,CaO \cdot Al_2O_3 + 6\,H_2O \rightarrow 3\,CaO \cdot Al_2O_3 \cdot 6\,H_2O. \qquad (1)$$

It likewise reacts quickly when calcium hydroxide is present, a substance which is split off in the hydration of the calcium silicates (alite and belite, see below):

$$3\,CaO \cdot Al_2O_3 + Ca(OH)_2 + 12\,H_2O \rightarrow 4\,CaO \cdot Al_2O_3 \cdot 13\,H_2O. \qquad (2)$$

Both these reactions would cause excessively rapid setting of the cement paste. Sulphate, in the form of gypsum or anhydrite-II, is therefore added as a retarder, interground with the clinker in the finish grinding mill. The hydration reaction in the presence of sulphate proceeds as follows:

$$\begin{array}{llll} 3\,CaO \cdot Al_2O_3 & + 3\,(CaSO_4 \cdot 2\,H_2O) & + 26\,H_2O & \rightarrow 3\,CaO \cdot Al_2O_3 \cdot 3\,CaSO_4 \cdot 32\,H_2O \\ C_3A & + 3\,(Cs \cdot 2\,H) & + 26\,H & \rightarrow C_3A \cdot 3\,Cs \cdot 32\,H \\ \text{aluminate} & + \text{gypsum} & + \text{water} & \rightarrow \text{ettringite/trisulphate} \\ \text{(1 volume)} & & & \quad \text{(8 volumes)} \end{array} \qquad (3)$$

The coarsely crystalline tabular calcium aluminate hydrates formed in the reactions (1) and (2) very quickly form a structure somewhat like a house of cards and possessing a certain amount of strength (corresponding to the "initial set" of the cement paste). On the other hand, reaction (3) – i.e., in the presence of sulphate – first produces finely crystalline ettringtite. This substance is deposited as a thin film on the surface of the cement particles in the first few hours of hydration. This film does not prevent the particles from sliding in relation to one another, i.e., the paste remains plastic. Only later, when the ettringite forms long needle-shaped crystals which bridge the water-filled spaces between the cement particles and enmesh the particles themselves, does the setting process begin (Fig. 15). The trisulphate (ettringite) subsequently undergoes transformation into monosulphate.

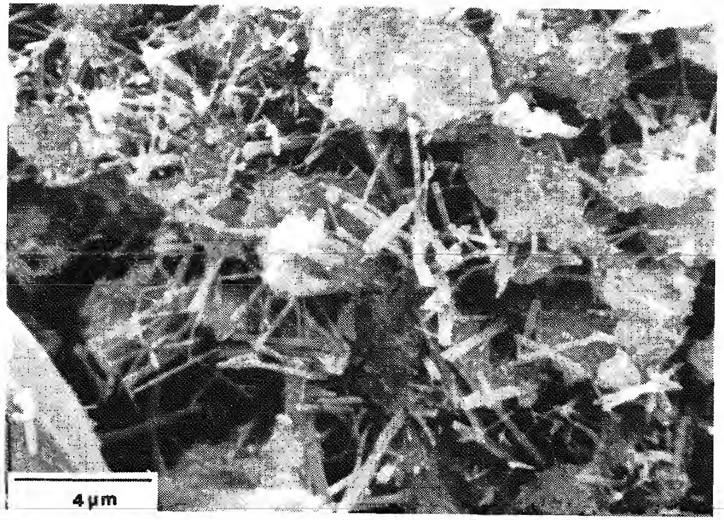


Fig. 15: Hardened cement paste with acicular ettringite crystals (scanning electron micrograph)

The sulphate content of the cement should be only so high that it is consumed in reaction (3) and not later than in the first 24 hours after mixing with water. Excess sulphate may, likewise in accordance with reaction (3), cause expansion phenomena in hardened mortar or concrete. Maximum permissible values of the SO_3 content are specified in order to prevent this (Table 7).

2.2 Alite

Alite (tricalcium silicate) reacts with water to form calcium silicate hydrates (CSH phases) containing less lime, while calcium hydroxide is split off. Belite (dicalcium silicate) shows similar behaviour. The hydration reaction is, for example:

$$\begin{array}{llll} 6\,(3\,CaO \cdot SiO_2) & +\,18\,H_2O & \rightarrow 5\,CaO \cdot 6\,SiO_2 \cdot 5\,H_2O & +\,13\,Ca(OH)_2. \\ 6\,C_3S & +\,18\,H & \rightarrow C_5S_6H_5 & +\,13\,CH \\ \text{alite} & +\,\text{water} & \rightarrow \text{CSH phase} & +\,\text{calcium hydroxide} \end{array} \quad (4)$$

The calcium silicate hydrates which are formed (Fig. 16) vary in the shape of their crystals (film-like, roll-like, fibre-like, etc.) and in their composition, depending on the conditions of formation (water-cement ratio, temperature, etc.). They are, however, always very fine-grained and are the principal strength-giving constituents of the hardened cement paste. Since the specific surface of the hardened paste is extremely high, namely, of the order of 3 000 000 cm^2/g (as compared with only about 3000 cm^2/g for cement), its strength is attributable to the co-operation of powerful adhesion forces (electrostatic forces of attraction acting between the exceedingly small hydrate phases) developed by the hydration products and the

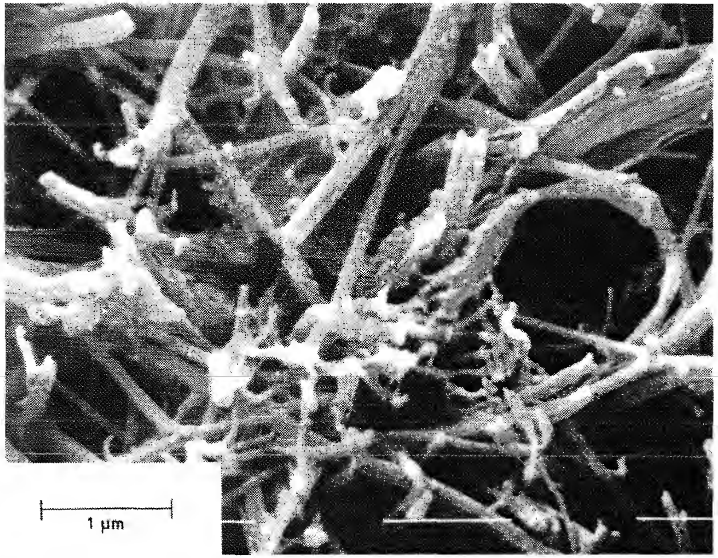


Fig. 16: Calcium silicate hydrates (CSH phases) in hardened cement paste (scanning electron micrograph)

Table 10: Heat of hydration of clinker phases (in J/g)

phase	heat of hydration	
	for reaction of individual phase	for reaction in clinker
C_3S	500	580
β-C_2S	250	350
C_3A	1350	1260
$C_2(A,F)$	420	160
MgO	850	
CaO	1160	

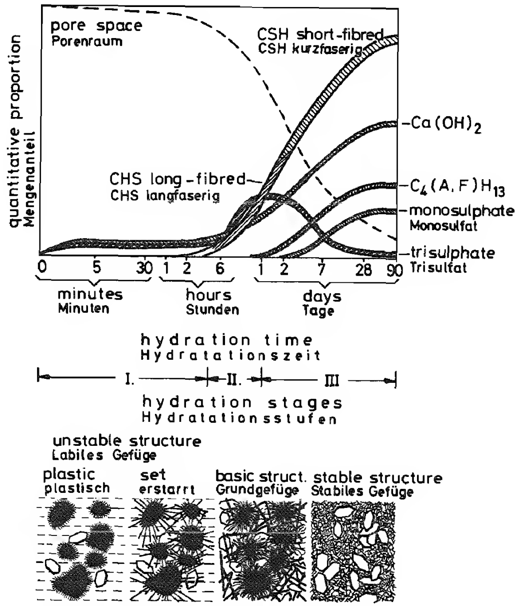


Fig. 17: Schematic diagram of the formation of the hydrate phases and the structure development in the hydration of cement (from Locher/Richartz/Sprung, 1976)

mechanical stabilization of the mass by interlacing of the newly formed compounds.
The calcium hydroxide which is formed in accordance with equation (4) produces a strongly basic environment ($pH > 12$) in the freshly hardened cement paste (and therefore in mortar and concrete). This high pH value inhibits the corrosion of embedded steel and is indeed what makes reinforced concrete such a durable material in which the reinforcing bars are normally so well and lastingly protected by the concrete. However, as a result of carbonation and other influences, this protective action may diminish in course of time.
Some indication of the respectively contributions of the clinker phases to the strength development of cement is given in Fig. 9. However, these results obtained for individual phases cannot be directly applied to the conditions actually occurring in cement paste, as is apparent also from the heat of hydration values given in Table 10. Fig. 17 schematically shows the sequence of formation of the hydrate phases and the structure development in the setting and hardening of portland cement.

3 Hydration of slag cements and pozzolanic cements

The hardening of cements consisting of portland cement clinker with blastfurnace slag or a pozzolanic material as the second major ingredient comprises two reaction subsystems. The portland cement reacts in the manner already described, while the interground ingredient is activated to undergo hydraulic hardening by the calcium hydroxide which is formed as a product of hydration of the calcium silicates alite and belite. The resulting reaction products of the hardening process are similar to those of portland cement, except that hardened slag cement contains less calcium hydroxide. These slag and pozzolanic cements moreover harden at a slower rate than portland cement and their rate of heat evolution is lower.

References

4, 6, 8, 13, 23, 28, 34, 38, 46, 52, 53, 56, 61, 62, 83, 84, 85, 86, 87, 88, 90, 91.

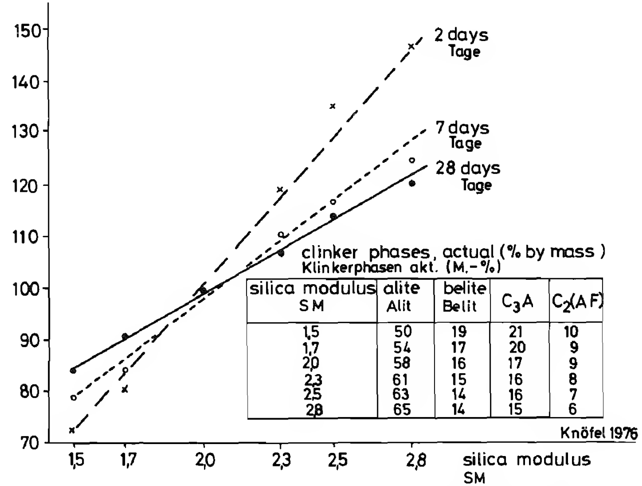
VIII. Relations between chemical reactions, phase content and strength of portland cement

It can reasonably be presumed that the chemical reaction pattern, the actual phase content and the strength of portland cement are at least loosely interassociated. For one thing, the new phases formed in the burning process (clinker phases) are dependent on the chemical character of the raw material. Furthermore, the strength-determining hydration products (hydration phases) are formed by reaction with water from the clinker phases.

Exactly definable relationships between the above-mentioned three properties or sets of properties – chemical reaction pattern, phase content, strength – can, however, at best be expected only if the following minimum conditions are fulfilled in the manufacture of the cement:

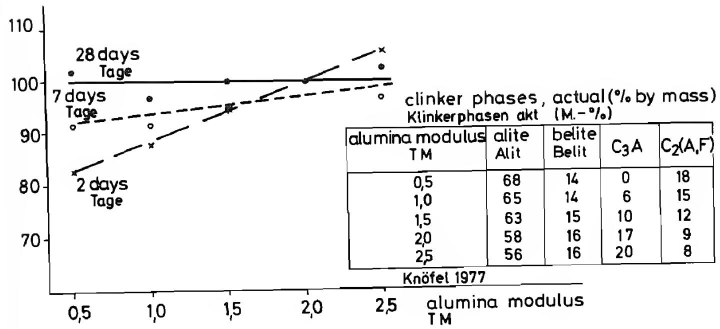
(1) adequate fineness and homogeneity of the raw meal;
(2) as a result: complete reaction of the meal to form clinker phases in the burning process;
(3) a clinker grinding process which produces equal reactive cement surface areas (specific surface values) for constant amounts of interground added sulphate.

The trends shown in Figs. 18, 19 and 20 are generally observed in industrial as well as in laboratory-made cements. The strength increase with increasing silica modulus is manifest, being more particularly due to the higher proportion of silicate in conjunction with lower proportions of aluminate and ferrite. The somewhat more marked increase in early strengths is attributable to the increase in alite (Fig. 18). Increasing the iron modulus (alumina ratio) only affects the early strength development as a result of the very considerable increase in aluminate, accompanied by a marked increase in heat of hydration which masks the decrease



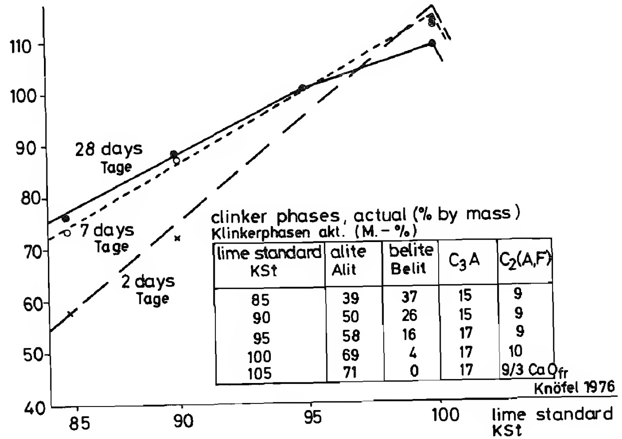
silica modulus SM	alite Alit	belite Belit	C_3A	$C_2(A,F)$
1,5	50	19	21	10
1,7	54	17	20	9
2,0	58	16	17	9
2,3	61	15	16	8
2,5	63	14	16	7
2,8	65	14	15	6

Fig. 18: Relative compressive strengths associated with variation of the silica modulus (laboratory cements; referred to cement with lime standard KSt I = 95; silica modulus 2.0; alumina modulus 2.0; 2.8% SO_3; fineness 3200 cm^2/g Blaine)



alumina modulus TM	alite Alit	belite Belit	C_3A	$C_2(A,F)$
0,5	68	14	0	18
1,0	65	14	6	15
1,5	63	15	10	12
2,0	58	16	17	9
2,5	56	16	20	8

Fig. 19: Relative compressive strengths associated with variation of the iron modulus (laboratory cements; referred to cement with lime standard KSt I = 95; silica modulus 2.0; alumina modulus 2.0; 2.8% SO_3; fineness 3200 cm^2/g Blaine)



lime standard KSt	alite Alit	belite Belit	C_3A	$C_2(A,F)$
85	39	37	15	9
90	50	26	15	9
95	58	16	17	9
100	69	4	17	10
105	71	0	17	9/3 CaO_{fr}

Fig. 20: Relative compressive strengths associated with variation of the lime standard (laboratory cements; referred to cement with lime standard KSt I = 95; silica modulus 2.0; alumina modulus 2.0; 2.8 SO_3; fineness 3200 cm^2/g Blaine)

in alite (Fig. 19). With an increase in the lime standard the compressive strength is notably increased, especially the early strength, the cause being the very large increase in alite content (Fig. 20).

From Figs. 18 to 20 it also emerges that the 28-day compressive strengths increase by about 10% as a result of raising the lime standard (by about 5 units) and the silica modulus (by about 0.3). Such an effect on strength cannot be obtained by varying the iron modulus. In general, it should be noted that the figures given are very approximate indications and are likely to vary greatly from one cement works to another.

The relative compressive strengths (referred to the respective 2-day strengths = 100) show the different amounts of hardening. With low silica modulus and iron modulus, as also with low lime standard, the subsequent hydration reactions still contribute a great deal to the strength attained. These diagrams, too, are merely approximate indications of trends.

These fundamentally clear-cut trends are liable to be considerably modified by the incorporation of subsidiary elements. The effect of MgO is shown in Fig. 22, and that of K_2SO_4 in Fig. 23, as examples. These effects, which are governed by the raw material characteristics, and also differences in the production conditions (raw material fineness and homogeneity, burning and cooling conditions, clinker grinding, cement storage) constitute a set of factors which make it impossible to make exact and reliable predictions of the strength development of cement by means of relatively simple calculations (formulas) based on the chemical reaction pattern or the phase content of the clinker concerned.

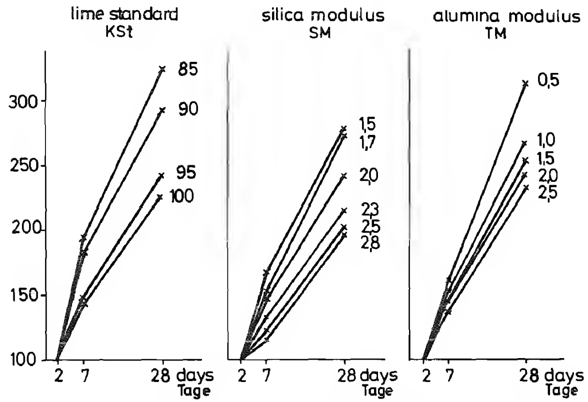
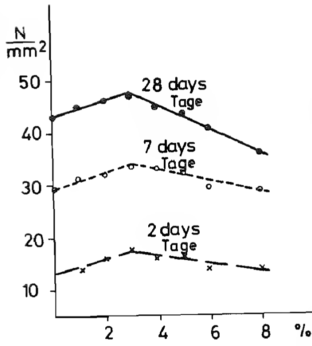


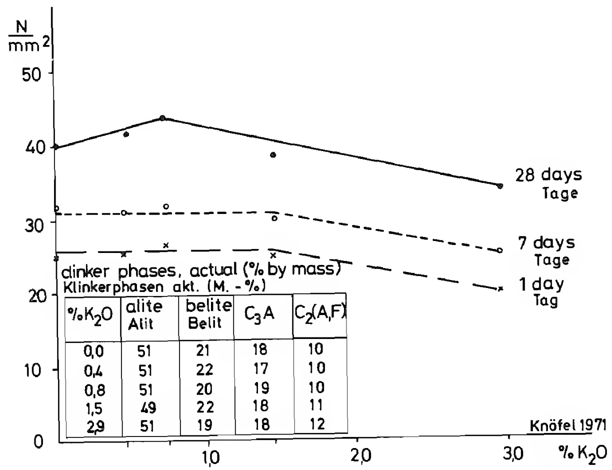
Fig. 21: Relative compressive strengths as a function of lime standard, silica modulus and iron modulus, referred to the respective 2-day strengths (laboratory cements)



clinker phases, actual (% by mass)
Klinkerphasen akt. (M.-%)

%MgO	alite Alit	belite Belit	C_3A	$C_2(A,F)$	periclase Periklas
0	59	15	17	9	0
1	62	11	15	12	0
2	64	9	13	14	0
3	67	6	11	16	<1
4	68	3	10	17	2
5	69	1	9	18	3
6,8	not determined nicht bestimmt				

Fig. 22: Effect of increasing MgO content on compressive strength development and clinker phase content (laboratory cements)



clinker phases, actual (% by mass)
Klinkerphasen akt. (M.-%)

$\%K_2O$	alite Alit	belite Belit	C_3A	$C_2(A,F)$
0,0	51	21	18	10
0,4	51	22	17	10
0,8	51	20	19	10
1,5	49	22	18	11
2,9	51	19	18	12

Fig. 23: Effect of K_2SO_4 on compressive strength development and clinker phase content (laboratory cements)

However, if the content of subsidiary elements and the production conditions can be kept approximately constant, as can usually be achieved in a particular cement works at least over a certain length of time, much more straightforward relationships will exist. In such cases the 28-day standard compressive strength of cement can be predicted with sufficient accuracy by means of a simple formula, provided that the three above-mentioned "minimum conditions" are complied with. One such formula is Knöfel's "strength index":

$$F\,28 = (3 \times \text{alite}) + (2 \times \text{belite}) + \text{aluminate} - \text{ferrite}.$$

Before this formula can be properly used, it is necessary to establish an appropriate correlation curve, obtained by plotting the strength index (F 28) against the compressive strength. For this purpose the phase contents should be determined quantitatively (by microscopic or X-ray examination) in at least ten cements (or clinkers) differing from one another as much as possible; the corresponding 28-day standard compressive strengths of these cements should also be determined. Then, with the aid of this correlation curve, the strength can be predicted by calculating the strength index from the quantitatively determined phase content. The validity of the correlation curves should be verified from time to time.

References

4, 7, 8, 23, 24, 28, 31, 33, 36, 39, 41, 46, 69, 71, 80, 83, 84.

IX. Types, strength classes, designation and quality control of cements

1 General

All cements are hydraulic binding agents, i.e., when mixed with water they will harden both in air and under water. The product of the hardening process – the "hardened cement paste" – is a water-resistant stone-like material.
As a general rule, cements of equal composition are more reactive in proportion as they are more finely ground and thus have a larger surface area at which the reactions can take place. Finer grinding tends to be associated with shorter setting times, higher early strengths and higher early rates of heat evolution (heat of hydration). It is in these respects that, for example, portland cement of class 35 differs from that of class 45.
The opposite trend (slower reaction, longer setting times, lower early strengths, lower heat of hydration) is associated with coarser grinding, higher belite content of the cement, and the addition of blastfurnace slag (slag cements) or pozzolana (pozzolanic cements, e.g., trass cement).
The effect of the above-mentioned influencing factors on the final strengths is small, however (see also Fig. 24).

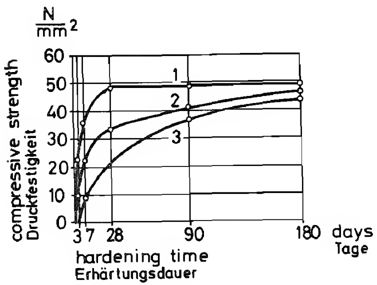


Fig. 24: Strength development of various cements (from Woods/Starke/Steinour, 1976): **1 = portland cement with 70% alite and 10% belite, 2 = portland blastfurnace cement with 60% slag, 3 = portland cement with 30% alite and 50% belite**

Table 11: Classification and designation of cements (from Cembureau, 1968)

symbol	special properties / designation
OC	Ordinary Portland Cement / normaler Portlandzement
RHC	Rapid-Hardening (or High Early Strength or High Initial Strength) Portland Cement / Portlandzement mit hoher Frühfestigkeit / schnellerhärtend
HSC	High Strength Portland Cement / Portlandzement mit hoher Festigkeit / hochfest
LHC	Low Heat (or Slow Hardening, Low Heat of Hydratation) Portland Cement, Medium Low Heat Portland Cement / Portlandzement mit niedriger Hydratationswärme
SRC	Sulphate-Resisting Portland Cement / Portlandzement mit hohem Sulfatwiderstand
AEC	Air-Entraining Portland Cement / Portlandzement mit Luftporenbildner
BL	Blastfurnace Cement / Hüttenzement
POZ	Pozzolanic Cement / Puzzolanzement

Note: The various types of cement can be further subdivided into classes (e.g.: OC I, OC II, Bl I, Bl II). The above subdivision for portland cement (according to properties) can also be applied to Bl and POZ.

Classification of cements can be based on various sets of criteria. Thus, the principal distinctive characteristics may be:

strength classes (minimum or average strengths; usually 28-day compressive strengths);
types of cement (portland cement, slag cement, pozzolanic cement);
important special properties (low heat of hydration, resistance to aggressive media, rapid strength development, etc.).

The main criterion of "strength class" is the basis of classification adopted in Standard DIN 1164 for cements in the Federal Republic of Germany (West Germany). The German Democratic Republic (East Germany) bases its TGL 28101/02 on "types of cement", while the American (USA) Standard ASTM C 150–76a and the classification of CEMBUREAU, Paris, are based on "important special properties" as the criterion. In each of these systems, the other criteria are employed for further subdividing the cements. The DIN 1164 classification will more particularly be considered here.

2 Classification and designation of cements

The strength classes listed in Table 12 are specified in DIN 1164. More particularly, the classification is based on the required minimum 28-day compressive strengths (determined by testing in accordance with DIN 1164, Part 7, see Section X). Besides, maximum permissible compressive strengths are laid down for the

Table 12: Strength classes (DIN 1164)

strength class		compressive strength in N/mm² at			
		2 days min.	7 days min.	28 days min.	max.
25[1]		–	10	25	45
35	L[2]	–	18	35	55
	F[2]	10	–		
45	L[2]	10	–	45	65
	F[2]	20	–	45	65
55		30	–	55	–

1 Only for cements with low heat of hydration and/or high sulphate resistance
2 Portland cement, Eisen portland cement, Hochofen cement and trass cement with slow early hardening behaviour are additionally given the symbol L, while the symbol F is added to cements with high early strength

cements Z 25, Z 35 and Z 45, and for this reason the cement manufacturers aim at achieving average strengths midway between the two specified limits for each class.
Cements Z 35 and Z 45 are furthermore subdivided according to their early hardening behaviour denoted by an appended letter:

cements with slow early hardening	L
cements with high early strength (rapid-hardening)	F.

The cements are produced by the intergrinding of portland cement clinker with a proportion of calcium sulphate (gypsum) to control the setting behaviour. In addition, the two German types of slag cement contain a substantial proportion of blastfurnace slag interground with the clinker, while trass cement similarly contains a substantial proportion of interground trass:

portland cement (made from portland cement clinker)	PZ
Eisenportland cement (containing at least 65% of portland cement clinker and not more than 35% of blastfurnace slag)	EPZ
Hochofen cement (containing 15 to 64% of portland cement clinker and 85 to 36% of blastfurnace slag)	HOZ
Trass cement (containing 60 to 80% of portland cement clinker and 40 to 20% of trass)	TrZ

(percentages by weight).

Furthermore, distinctions are based on special properties:

cements with low heat of hydration (maximum heat of hydration after 7 days: 270 J/g)	NW
cements with high sulphate resistance (two types: PZ with ≤ 3% C_3A, potential according to Bogue, and ≤ 5% Al_2O_3; HOZ with ≥ 70% blastfurnace slag)	HS
cements with low effective alkali content (not standardized) (maximum total alkali content in Na_2O equivalent: ≤ 0.60% in PZ; ≤ 0.90% in HOZ with > 50% slag ≤ 1.10)	NA

(percentages by weight).

The complete standard designation of a cement comprises its indication of strength class, cement type and special properties (if any). Examples:

(1) A portland cement (PZ) with a 28-day minimum compressive strength of 35 N/mm² (35) and 2-day minimum compressive strength of 10 N/mm² (F):
designation according to DIN 1164: PZ 35 F.
(2) A Hochofen cement (HOZ) with a 28-day minimum compressive strength of 35 N/mm² (35), a 7-day minimum compressive strength of 18 N/mm² (L), and high sulphate resistance (HS):
designation according to DIN 1164: HOZ 35 L-HS.

Other standard cements complying with DIN 1164 are special cements such as white cement, water-repellent (hydrophobic) cement and highway engineering cement.

Table 13: Chemical composition of cements (reference values, in % by weight)

oxide	portland cement: ordinary	portland cement: rapid-hardening	portland cement: with high sulphate resistance	portland cement: white	Eisen portland cement	Hochofen cement	trass cement	oil shale cement
CaO	61–69	66	64	67	50–66	44–61	47–60	54–61
SiO_2	18–24	20	21	23	19–28	21–30	20–28	18–23
Al_2O_3	4–8	5	4	4	4–12	6–15	5–10	6–8
Fe_2O_3	1–8	3	7	<1	1–4	1–3	2–4	3–4
MgO	<5	<2	<2	<2	<6	<7	<3	<2
SO_3	2–4	3–4	2–4	2–4	2–4	2–4	2–4	2–4

Oil shale cement and trass blastfurnace slag cement are permitted under special certificate of approval in the Federal Republic of Germany, but are not standardized.

3 Constituents of cements

The principal constituents of the above-mentioned cements are portland cement clinker, blastfurnace slag and trass (see Sections IV and V).
The content of magnesium oxide (MgO), referred to the ignited portland cement clinker, is not allowed to exceed 5% by weight, while the sulphate content (as SO_3) must comply with the values given in Table 7. Other admixtures in amounts up to 1% by weight are permitted, provided that they do not promote corrosion of reinforcing steel. Chlorides (Cl^-) are not allowed to be added to cement; the inherent Cl^- content from the raw materials must not exceed 0.10% by weight.
Determination of the chemical composition of cements should be done in accordance with DIN 1164, Part 3. Table 13 gives some approximate guiding values for the chemical composition.

4 Supply and identification of cements

"Cement is allowed to be put only in transport containers which are clean and free from residues of earlier deliveries. It must not become contaminated in transit" (DIN 1164, Part 1).
Delivery notes for bulk cement or labels on sacks should give the following information: type of cement, strength class, designation of special properties (if any), name of supplying works, gross weight of sack or net weight of bulk cement, quality control indication. Delivery notes for cement supplied in bulk should furthermore state: date and time of delivery, vehicle registration number, name of customer, order number and consignee.
In addition, distinctive colour identification for strength class should be displayed on the cement sacks (Table 14). In the case of bulk cement delivery a distinctively coloured weatherproof sheet (size DIN A5, colour and lettering conforming to

Table 14: Distinctive colours for the strength classes (DIN 1164)

strength class		distinctive colour	colour of lettering
25		violet	black
35	L	light brown	black
	F		red
45	L	green	black
	F		red
55		red	black

Table 14) for affixing to the storage bin should accompany the delivery note. The information printed on this coloured sheet should comprise: type of cement, strength class, designation of special properties (if any), name of supplying works, quality control mark and delivery date stamp.

5 Quality control

Due conformity to the cement quality requirements of DIN 1164 (composition and properties) should be verified and monitored by quality control ("internal" control by the cement manufacturer and "external" control by an authorized independent supervisory organization, DIN 1164, Part 2).

5.1 Internal quality control

"So long as a cement is being manufactured and in so far as a limiting value is specified in DIN 1164, Part 1, the cement manufacturer must test the composition and the properties of each type of cement and strength class in the cement works. The following are to be tested at least once a day:

setting
soundness.

At least twice a week:

loss on ignition
content of carbon dioxide CO_2
insoluble residue
content of sulphate SO_3
fineness of grinding
compressive strength at each specified age (see DIN 1164, Part 1).

At least once a month:

principal constituents of the cement
heat of hydration
the composition required to ensure high sulphate resistance.

The results of the internal quality control should be recorded in writing and, if possible, statistically analysed. The records should be kept for at least five years and be made available to the supervisory organization (external quality control) on request" (DIN 1164, Part 2).

5.2 External quality control

External quality control is as a rule performed by an officially recognized quality control organization; at present this is the German Cement Works Association (Verein Deutscher Zementwerke), Düsseldorf.
The supervisory (external quality control) organization should monitor the cement works' own internal quality control, primarily by inspection of the relevant records and documents. In addition, the supervisory organization should, in order to verify

compliance with the conditions laid down in DIN 1164, Part 1, carry out the following tests for each cement type and strength class in current production:

At least once in every two months:

loss on ignition, content of carbon dioxide CO_2, insoluble residue, content of chloride, fineness of grinding, setting, soundness, compressive strength at each specified standard age, principal constituents of the cements.

At least once every six months:

heat of hydration, the composition required to ensure high sulphate resistance.

A test report should be made. If the cement is found to fulfil the requirements of DIN 1164, the packaging and delivery note is allowed to carry the inscription "Quality controlled in conformity with DIN 1164" and the sign or mark of the quality control organization (e.g., "VDZ") (Fig. 25).

Fig. 25: Quality mark (left) and mark of the quality control institution (Verein Deutscher Zementwerke, Düsseldorf, right) (DIN 1164)

According to the "Technical guarantee conditions for standard cements" the user of the cement does not have to carry out any checking or monitoring of the standard values. However, as a precaution against any guarantee claims, it is essential that a sample of each cement consignment be kept for possible future reference. This sample should be properly representative of the consignment (average sample), have a weight of at least 5 kg, be stored dry and under airtight conditions, and be unmistakably labelled (time and date of delivery, name of supplying works, type and strength class of the cement, No. of delivery note).

6 Suggestions for the use of cements with reference to their general and special properties (from: Zement-Merkblatt, issued by Bundesverband der Deutschen Zementindustrie)

Portland cements, Eisen portland cements, Hochofen cements and trass cements are so classified in DIN 1164 that their properties are, in the main, characterized by their standard designations.

Z 25: Cement with very slow strength development and heat evolution, designated by NW (low heat). If this cement has a high resistance to sulphate

attack, it is additionally given the designation HS. In general, this class of cement is used in mass concrete.

Z 35 L: Cements with the same 28-day strength as Z 35 F, but slower early hardening and therefore correspondingly longer formwork stripping times, good subsequent strength development. Because of low heat evolution this cement is especially suitable for massive structural members.

Z 35 F: Cements with normal early hardening and medium heat evolution.

Z 45 L: Cements with the same 28-day strength as Z 45 F, but with lower 2-day strengths.

Z 45 F: Cements with high early strength (rapid-hardening), so that early formwork stripping is possible. Because of rapid strength development and rate of heat evolution, suitable for precast concrete and for winter construction.

Z 55: Cements with very high 2-day and 28-day strengths, for cases where high early concrete strength is needed and for very high-strength concrete construction. These cements are especially suitable for concrete to be placed at low temperatures, so that resistance to freezing is attained as quickly as possible.

The special additional properties of low heat evolution (NW) and high sulphate resistance (HS) have already been mentioned above (see IX.2 and IV.2), as have also certain special cements, more particularly: white cement (PZ 45 F with low iron oxide content) and water-repellent (hydrophobic) cement (insensitive to moisture; reacts with water only after intensive mixing; available in strength classes Z 35 F and Z 45 F).
Different cements should not be mixed with one another, certainly not on the construction site, as the facilities for uniform blending are not available there. Otherwise, for example, variations in colour are liable to occur, "special" properties of the cements may be impaired, etc. If mixing of different cements is unavoidable, however, then only the properties and values of the cement with the lower cement class should be adopted for the resulting mixture. Quick setting will occur when a mixture of high-alumina cement with a standard cement conforming to DIN 1164 (PZ, EPZ, HOZ, TrZ) is used for making mortar or concrete.

References

4, 8, 11, 13, 14, 16, 21, 23, 28, 83.

X. Cement testing

The standard test requirements for cements used in the Federal Republic of Germany are specified in DIN 1164, Parts 3 to 8. The tests relate to the determination of the following: composition (Part 3), fineness (Part 4), setting times (Part 5), soundness (Part 6), strength (Part 7) and heat of hydration (Part 8).

It would be outside the present scope to deal with the determination of the composition of cements, more particularly the chemical analysis. As for the other properties, the procedures will be briefly outlined. For further details the relevant parts of the Standard will have to be consulted.

1 Fineness (DIN 1164, Part 4)

As specified in Part 1 of DIN 1164, a cement conforming to this Standard must not leave a residue of more than 3% by weight on the 0.2 mm test sieve (DIN 4188). The specific surface determined by the air permeability method should be not less than 2200 cm²/g (in special cases not less than 2000 cm²/g).

1.1 Sieve residue

The content of coarse particles is determined as the residue retained on the test sieve with 0.2 mm aperture size (DIN 4188, Sheet 1) by manual or mechanical sieving. The sample for the sieve test should consist of 100 ± 0.100 g of dry cement. Sieving is stopped when the residue does not decrease by more than 0.1% on continuation of sieving for a further 2 minutes. The amount retained on the sieve is stated in % by weight, referred to the initial sample.

1.2 Specific surface

"The specific surface of cement in cm²/g is calculated from the air permeability of a bed of cement, its porosity, the density of the cement and the viscosity of air. The measure of the permeability is the time it takes for a certain quantity of air to flow through the bed under specified conditions" (DIN 1164, Part 4).
For performing the test a predetermined quantity of cement is put into the standardized apparatus and is gently compacted to a predetermined volume. Then air is drawn through the bed of cement by suction produced by a falling column of liquid. The time it takes for the level of the liquid in the U-tube of the apparatus to fall a certain marked distance is measured. This time is a measure of the fineness of the cement: the finer it is, the longer will it take for the air to flow through the bed, and vice versa. The specific surface is calculated from:

$$O_{sp} = \frac{K \cdot \sqrt{e^3} \cdot \sqrt{t}}{\rho\ (1-e) \cdot \sqrt{10\eta}}$$

where:
- O_{sp} specific surface in cm²/g
- e porosity in parts by volume
- t time of air flow in seconds
- ρ specific gravity of the cement
- η dynamic viscosity of the air in Pa · s
- K apparatus constant.

2 Setting times (DIN 1164, Part 5)

Obviously, in order to allow sufficient time for applying the mortar or placing the concrete, cement must not set too quickly. According to DIN 1164, Part 1, standard cements must not begin to set earlier than 1 hour after mixing, and setting must be completed not more than 12 hours after mixing.
The setting times (initial and final set) are determined with Vicat's needle apparatus on a neat cement paste: The cement is passed through a 1 mm test sieve and a quantity of 500 g is mixed with 25–30% (by weight) of water – depending on the type of cement concerned – in a standard two-speed mixer for a total of 3 minutes. A certain "standard" consistency of the cement paste must be attained by variation of the amount of mixing water to suit the cement under investigation. This consistency is ascertained by putting the cement paste in a mould consisting of an ebonite ring on a sheet of glass and by determining the penetration depth of a "plunger" applied to the top surface of the cement paste specimen. When the latter has attained the standard consistency (ascertainable by trial and error with varying amounts of mixing water, if necessary), the initial and the final setting time can be determined with the "needle" of the Vicat apparatus. The initial set is considered to occur when the needle penetrates to a distance of 3 to 5 cm from the bottom of the mould, i. e., remains stuck in the paste at this distance above the glass sheet. For determining the final set, the mould with the sample is removed from the glass and replaced in the reversed position. The final set is considered to occur when the needle penetrates not more than 1 mm into the sample. In both tests, i. e., for initial and for final setting time, the penetration of the needle should be measured at 10-minute intervals.

3 Soundness (DIN 1164, Part 6)

By "soundness" is understood the ability of the cement to maintain a constant volume. Thus, a cement is to be rated as sound if, after it has hardened, it remains free from expansion effects which may crack, loosen and destroy the hardened paste.
Unsoundness, i. e., lack of volume stability, is caused by a high content of free MgO, causing magnesia expansion (for this reason DIN 1164 specifies that the MgO content must not exceed 5.0% by weight), by excess sulphate, causing sulphate expansion (DIN 1164 specifies an upper limit for the SO_3 content: see Table 7), and by substantial amounts of free CaO (uncombined lime, causing lime expansion; this is monitored by the boiling test). For reactions see Section VII.2.
The test for soundness specified in DIN 1164, Part 6, is the boiling test and is performed at the same time as the setting test, using surplus cement paste (or otherwise paste of the same consistency prepared for the purpose). Half this sample is formed into a lump, placed on the centre of a sheet of glass, and gently vibrated, so that it spreads into a "pat" about 10 cm in diameter and 1 cm thick, which is allowed to set and harden at ≧ 90% relative humidity for 24 hours. The sample is then boiled for 2 hours. If cracking or warping occurs, the cement must be rated as having failed the test. But if the sample remains sharp-edged, free from

cracks and not greatly distorted (> 2 mm), it has satisfied the test, i. e., is "sound". The more stringent autoclave test in accordance with ASTM C 151-76a is a criterion for magnesia expansion as well as the expansion due to too much free lime.

4 Strength (DIN 1164, Part 7)

Depending on its strength class, a cement should attain the compressive strengths listed in Table 12. These values are averages obtained from tests on six test prism halves. The test procedure is known as the ISO-RILEM-CEM method.
It is performed on mortar prisms with dimensions of 4 cm × 4 cm × 16 cm. The mortar consists of a mixture of cement, standard sand (comprising particle fractions of 0.08–0.5 mm, 0.5–1.0 mm and 1.0–2.0 mm, in equal parts) and water in the proportions of 1 : 3 : 0.5 (quantities for making three prisms are 450 g of cement, 1350 g of sand, 225 g of water). It is mixed in a two-speed standardized mixer, put into steel moulds and compacted on a vibrating table. The moulds containing the mortar prisms are stored at ≧ 90% relative humidity for 1 day, then the prisms are carefully demoulded and kept in water at 20° ± 1° C up to the time of testing.
The flexural strength should be determined by fracturing at least three prism specimens in the middle by means of an apparatus specified in the Standard.
The compressive strength is determined immediately afterwards on the six halves of the prisms fractured in the flexural test. The compression testing machine conforming to DIN 1164 has to satisfy strict requirements. To comply with Part 1 of this Standard, only the compressive strength need be determined.

5 Heat of hydration (DIN 1164, Part 8)

Cements with the special property "low heat of hydration" (designation NW) are not allowed to evolve more than 270 J of heat per gramme of cement in the first 7 days after mixing with water. They are used more particularly for mass concrete structures which, if made with ordinary cement, might undergo an excessive rise in temperature causing stresses and cracking. Low-heat cements are chiefly portland cements with a high content of dicalcium silicate (and lower content of tricalcium

Table 15: Heat of hydration of cements (reference values)

cement	heat of hydration (J/g) at 7 days	on complete hydration
portland cement	–	380–525
Eisen/Hochofen cement	–	360–440
trass cement	–	340–420
low-heat cement	< 270	–

silicate and/or tricalcium aluminate) and slag cements with a high content of blastfurnace slag. Values for the heat of hydration of cements are given in Table 15.
In accordance with DIN 1164, Part 8, the heat of hydration is determined with a heat-of-solution calorimeter. As stated there, "... this method is intended for the determination of the specific heat in J/g that is released when a cement undergoes hydration under isothermal conditions. The heat of solution of the unhydrated cement sample as well as that of the sample hydrated at 20° C (water-cement ratio w/c = 0.4) in a specified acid mixture is measured. The difference between the two heats of solution is the heat of hydration."
The test apparatus comprises a heat-of-solution calorimeter with accessories (Dewar flask, stirrer, funnel, etc.), an officially calibrated Beckmann thermometer and an appropriate acid mixture (nitric acid + hydrofluoric acid). The cement paste samples (their mix proportions, mixing procedure and temperature are specified) are stored in a water bath at 20° ± 0.5° C. The heats of solution of the unhydrated and of the hydrated cement are determined from the rise in temperature occurring when the samples dissolve (the test should be performed in constant-temperature surroundings) and from the determinations of the CaO content (or the losses on ignition, if applicable). Formulas for calculating the heat of solution from the test data are given in the Standard. It is an elaborate procedure.

References

3, 4, 8, 10, 11, 13, 14, 15, 28, 46, 75, 83.

Cement Standards of various countries
(For symbols and designations see Table 11)

Type of cement	Standard	Issuing authority
Federal Republic of Germany		
OC, HSC, SRC/LHC BI, POZ	DIN 1164 Nov 1978	DIN Deutsches Institut für Normung, Burggrafenstrasse 4–7, D-1000 Berlin 30
German Democratic Republic		
OC, HSC, SRC/LHC BI, POZ	TGL 28101/01 TGL 28101/02	Amt für Standardisierung, Abt. Dokumentation, Mohrenstrasse 37a, DDR-1026 Berlin
France		
all cements	NF P 15-300 NF P 15-301 1978 edition	Association Française de Normalisation, Tour Europe, Cedex 7, F-92080 Paris la Défense
Great Britain		
OC, RHC	BS 12: 1978	British Standards Institution British Standards House, 2 Park Street, London W.1
LHC	BS 12: Part 2: 1974	
SRC	BS 4027. Part 2· 1972	
United States of America		
OC, SRC, LHC, RHC AEC	ASTM C 150-78a	American Society for Testing and Materials, 1916 Race Street, Philadelphia, Pa. 19103
BL, POZ	ASTM C 595-76	

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D. Manufacture of cement

I. Materials preparation technology

By H. Schneider and U. Binder

I. Materials preparation technology

The purpose of the preparatory processing of the raw materials is to convert these chemically and mineralogically different materials, usually supplied to the plant in coarse lumps, into raw meal or slurry of homogeneous composition. This has to be accomplished with suitably chosen machinery and methods, and at lowest possible cost, in order thus to fulfil the basic conditions for an economical burning process.

Primary crushing, pre-blending, drying, grinding, combined grinding and drying, and homogenizing are the principal processing stages in the preparation of the raw materials for cement manufacture. Screening and classifying are separating methods which are used in the cement industry in order to carry out the size reduction operations with greater economy. On the other hand, beneficiation of raw materials by separation of unutilizable constituents and concentration of the utilizable ones by screening or classifying is only exceptionally applied in cement manufacture. Limestone and clay – the two principal raw materials for cement clinker production – as well as secondary raw materials containing aluminium oxide, silicon oxide and iron oxide, which are used as admixtures in the process, are almost everywhere available in adequate quantities and suitable chemical composition.

Elaborate beneficiation treatments such as are widely employed in ore and coal preparation are therefore generally not required in the cement industry and, apart from certain individual cases, not economically viable either.

1 Primary reduction

Reducing the raw materials to a fine powder – conventionally called "meal" – is necessary in order to produce a homogeneous mixture which will quickly be converted in the kiln into a homogeneous clinker containing no free lime.

As a rule, size reduction (comminution) of the raw materials is effected in at least two main stages: crushing (primary reduction) and grinding (fine reduction). In the cement industry it is not usual to make a sharp distinction between these stages in terms of particular product sizes of the crushing machinery. Indeed, the borderline is variable, depending on the performance and attainable reduction ratio of the crushers and grinding mills and thus depending also on the technical development of these machines.

Generally speaking, crushing denotes the size reduction process that breaks down the material to a particle size suitable as feed for the next main stage, i.e., grinding. In applying the distinction between crushing and grinding it is of no consequence whether either or both of these main stages are accomplished in one or more individual stages.

In present-day cement manufacture, with due regard to the possibilities of the reduction machinery employed, crushing is taken to mean reducing the particle size to between about 80 and 20 mm. This crushed product is further reduced by grinding to a fineness below about 0.2 mm size, in which condition it is called raw meal and is ready for feeding to the kiln.

1.1 Definitions and characteristics

Before describing the actual size reduction processes it is necessary to define some commonly used concepts associated with them:
Single-stage reduction means that the material is reduced to the desired product fineness by the action of just one machine, which may operate either on the open-circuit (single-pass) or the closed-circuit principle (Fig. 1). **Multi-stage reduction** is effected in two or more machines in series, each of which constitutes one stage of the reduction process and which operates either in open or in closed circuit.

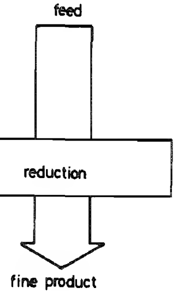


Fig. 1: Open-circuit size reduction

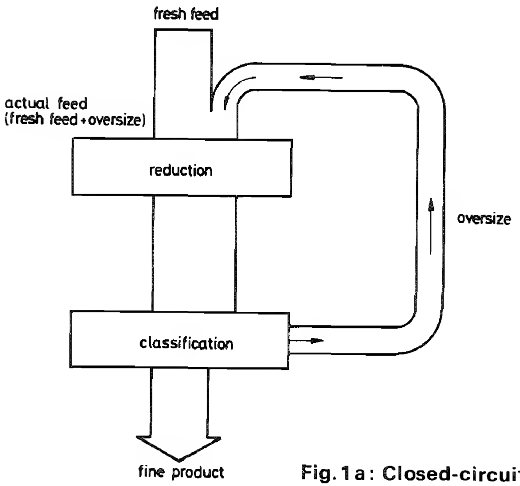


Fig. 1a: Closed-circuit size reduction

In **open-circuit reduction** the material passes only once through the machine, whereas in **closed-circuit reduction** the material discharged from the machine is separated by screening or classifying (the latter usually by air separation) into fine and coarse particles, the latter being returned to the machine for further reduction (Fig. 1 a).
The so-called **reduction ratio** is frequently applied as a criterion for judging the operating range of crushers. It is the ratio of the size (linear edge dimension) of the largest piece in the feed material to the size of the largest piece in the crushed product. As it is difficult actually to determine the largest sizes in the feed and in the product, these respective particle sizes are instead defined in terms of a certain percentage (by weight) passing a screen, e.g., 95% or 80% or 63.2% (Fig. 2).

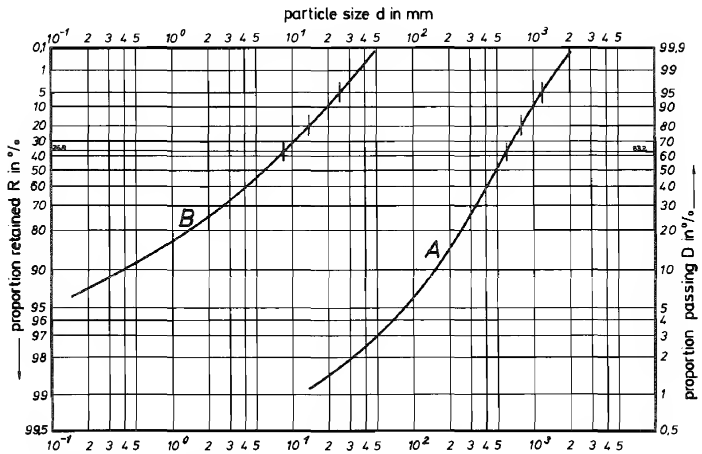


Fig. 2: Reduction ratio "Z"

$$Z_{95} = \frac{1200}{25} = 48;\ Z_{80} = \frac{800}{14} = 57,\ Z_{63.2} = \frac{600}{8.6} = 70$$

A = crusher feed (fragmented rock obtained by blasting)
B = crusher product (discharged from a single-rotor hammer crusher)

The **granulometric composition** of the feed or the product of a size reduction machine is determined by screening or sieving in the coarse particle size range (above about 50 microns).
The result of the particle size analysis (screen or sieve analysis) can be represented in a numerical table and/or as a **particle size cumulative distribution curve**

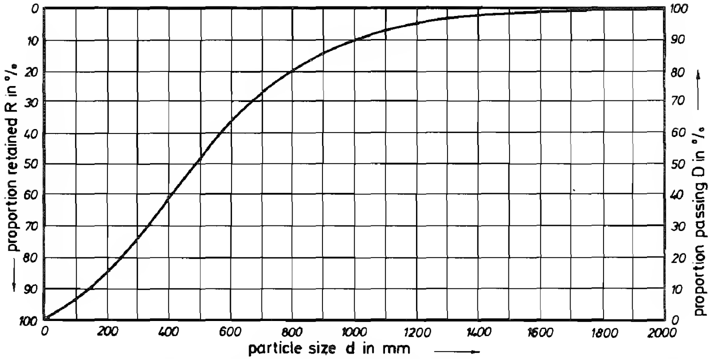


Fig. 3: Cumulative distribution curve for rock pile produced by large-hole blasting (linear scales on both axes)

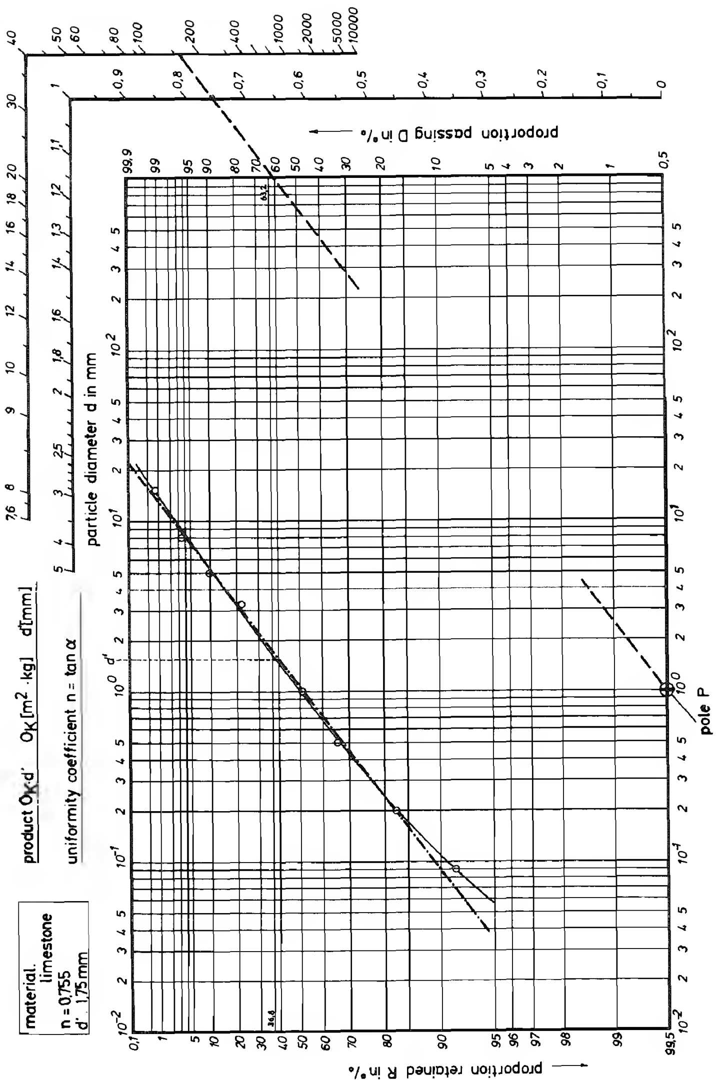
from which the percentage retained on or passing any particular aperture size can be read (**Fig. 3**). In the diagram the particle sizes are shown on the horizontal axis, while the percentages (by weight) are marked on the vertical axis. A linear scale may be adopted for both axes or, alternatively, only for the vertical axis, while the particles are plotted to a logarithmic scale.

Quite often the well-known Rosin-Rammler-Bennett (RRB) particle size distribution is assumed for comminuted materials, which uses log d for the horizontal and $\log\left(\log \frac{100}{R}\right)$ for the vertical axis (d = particle size, R = percentage retained).

If the exponential relationship established by Rosin, Rammler and Sperling is strictly conformed to, the distribution curve appears as a straight line which is characterized by two values. the equivalent particle size d' and the uniformity coefficient n (Fig. 4), where d' is the size corresponding to 36.8% (by weight) retained as residue on the sieve (oversize) and n is the tangent of the slope of the line. Particle size distribution diagrams are commercially available which are provided with scales on the vertical and horizontal axes enabling the values of n and of the specific surface of comminuted materials to be read.

The actual values determined in tests generally deviate more or less from the theoretical straight lines. Even so, the exponential relationship is a useful approximate equation.

Fig. 4: Curve for oversize particles in hammer mill product (RRB diagram). Material· limestone



The logarithmic division of the horizontal axis of a particle size distribution diagram offers the advantage that the finer sizes are characterized more prominently, which is appropriate because of their greater importance than the coarser ones in determining the overall surface area of the comminuted material concerned.
By differentiation the **particle size distribution diagram** can be derived from the cumulative distribution curve (Fig. 5): The horizontal axis is divided into equal portions, each representing a particle size class or fraction, and for each portion the corresponding ordinate is determined, indicating the percentage (by weight) of this size class in the comminuted material as a whole.

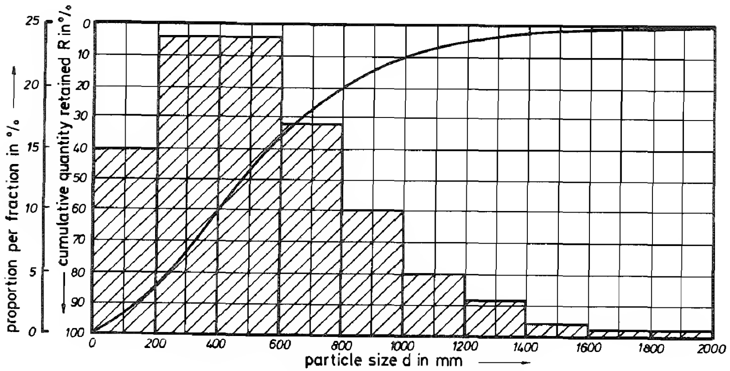


Fig. 5: Particle size distribution diagram for rock pile produced by large-hole blasting

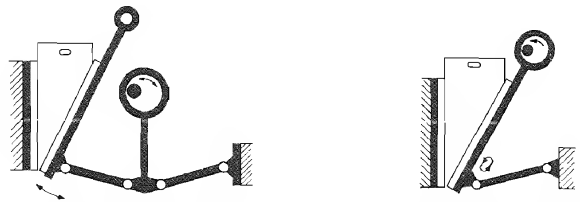
1.2 Types of crusher

Solid rock which has been dislodged from its natural deposit by blasting or ripping forms a coarsely fragmented rock pile, initially with its natural inherent moisture. The hardness, fragment size, moisture content, plasticity and abrasiveness of the material are important factors affecting the choice of the size reduction machines and methods for dealing with it. Hard materials causing severe abrasive wear are reduced with slow-running machines such as jaw crushers and gyratory crushers, which function by developing mainly a compressive action. For medium-hard to hard materials impact crushers and hammer crushers are more suitable, they achieve size reduction mainly by impact.

1.2.1 Jaw crushers

Jaw crushers are used for the primary reduction of very hard and abrasive admixtures for cement manufacture, such as quartzite or iron ore, and of large

lumps of clinker or kiln coating (as discharged from rotary coolers or shaft kilns, for example). On the other hand, they are seldom used for the crushing of limestone in cement works.
The reciprocating motion of the crushing jaw of the double-toggle (or Blake type) jaw crusher subjects the material to a mainly compressive action. This machine is especially suitable for crushing very hard material fed in coarse lumps (Fig. 6a)
In the single-toggle jaw crusher the jaw moves not only backwards and forwards but also up and down, so that there is attrition as well as compressive crushing action. Crushers of this type are more suitable for the reduction of hard to medium-hard material fed in smaller lumps (Fig. 6b)
Jaw crushers are sensitive to moist and plastic feed material and tend to choke if there is a substantial proportion of fine particles in the feed. The attainable reduction ratio is between about 6:1 and 8:1. For obtaining a product of favourable size for feeding to grinding mills it is generally necessary to apply secondary crushing in another type of crusher.
The particle size distribution of the jaw crusher product is considerably affected by the loading of the machine; a crusher operating substantially below capacity will yield a coarse product with a high proportion of oversize.



(a) double-toggle crusher   (b) single-toggle crusher

Fig. 6: Jaw crushers

1.2.2 Gyratory crushers

The gyratory crusher is seldom found in the European cement industry, which uses mainly medium-hard to hard and not very abrasive limestone. In other parts of the world, however, it is more commonly employed in the industry. Its size reduction is achieved mainly by compressive action between the fixed conical bowl and the oscillating cone-shaped crushing head, which functions somewhat like a pestle in a mortar. The lower end of the shaft carrying the crushing head is mounted in an eccentric which performs a horizontal rotary motion, while the upper end is mounted in a fixed ball-and-socket type bearing. As in the jaw crusher, the width of the crushing gap continually varies between a maximum and a minimum.
The width of the gap can be altered by raising or lowering the crushing head, an adjustment that takes only a few minutes to perform and is effected mechanically or

(in machines of more modern design) hydraulically. Increase in gap width due to wear of the bowl and crushing head can thus be compensated, so that the service life of these parts can be extended by some 50 to 60% without having to recondition or replace them, while the product size remains approximately unchanged throughout their lifetime. Vertical adjustment of the head in relation to the bowl enables the gap width to be varied within a range of 15–20% from the average setting (Fig. 7).
The ratio between the radial width A of the feed opening and the maximum discharge gap width C in large primary crushers is between 6:1 and 9:1, which corresponds to the attainable reduction ratio for predominantly cubic material. If the machine is fed with material of a more irregular shape, this ratio, referred to the maximum dimensions of the pieces, may be as high as 12:1 to 15:1.
The largest gyratory crushers in current use attain throughputs of over 6000 t/hour and have feed openings 1500 mm × 4400 mm in size (A × B), while the discharge gaps range in width from 150 to 250 mm.
A jaw crusher designed for a certain throughput rate can accept larger pieces of rock than the normal gyratory crusher. In order to cope with equally large-sized feed, the gyratory crusher has to be over-designed in terms of capacity.
In a special form of the machine, called the unifeed gyratory crusher (Krupp-Esch; Morgardshammar), this drawback has been eliminated. It is substantially similar to an ordinary gyratory crusher, except that the feed opening is provided with an enlarged receiving space on one side, which functions as a pre-crushing chamber (Fig. 8). The oscillating motion of the crushing head is similar to that in an ordinary gyratory crusher.
A general advantage of the gyratory crusher is that it is unaffected by overloading. It requires no special feeding device. the fragmented rock coming from the quarry

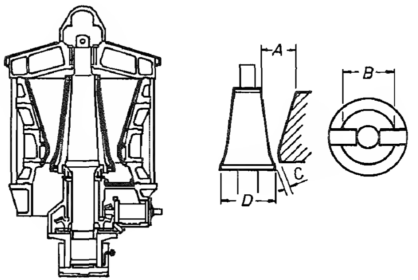


Fig. 7: Gyratory crusher

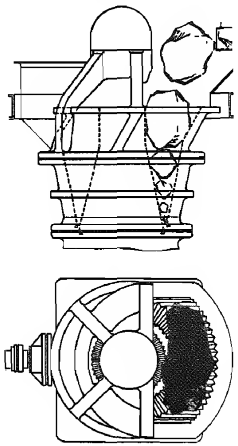


Fig. 8: Unifeed gyratory crusher (Krupp-Esch)

or stockpile in heavy trucks can be tipped straight into the feed opening. Uniform size distribution in the crushed product can, however, be obtained only if a controlled rate of feed is maintained.
Like the jaw crusher, the gyratory crusher is sensitive to moist and plastic feed material and it tends to choke if the material has a high fines content.

1.2.3 Roll crushers

Roll crushers are used for the primary reduction of medium-hard moist and abrasive materials such as marl, shale and clay (Fig. 9). The feed is subjected to compressive and shearing action between a pair of counter-rotating rolls, which may be either smooth or corrugated or provided with tooth-like projections. The teeth give better bite to the feed and concentrate the action of the crushing force, enabling large and compact pieces of rock to be split.

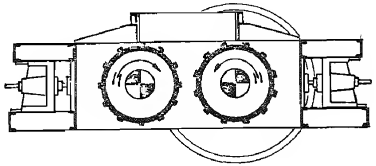


Fig. 9: Double-roll crusher

For primary reduction the width of the rolls is approximately equal to their diameter, the ratio of these dimensions usually being within the range of 0.8 to 1.2. The attainable size reduction ratio is fairly low, only from about 3:1 to 5:1. Circumferential velocities of the rolls are 5–9 m/second.
Double-roll (or twin-roll) crushers with 1800 mm roll diameter and approximately equal effective roll width attain throughputs of 1000–1200 t/hour for a gap width of about 250 mm between the rolls and can accept feed material up to 1000 mm in size.
In some machines the two bearings of one crushing roll are fixed to the frame of the crusher, while those of the other roll are mounted on slide rails. This movable roll is held in its predetermined working position with the aid of pull-rods and springs.
The movable mounting enables the crushing gap width to be varied, while the springs provide some "give" to allow uncrushable foreign bodies in the feed to pass. Double-roll crushers in which both rolls are movably mounted are also available.
As a rule, the two rolls are driven separately, each through a V-belt drive. The specific power consumption is in the range of 0.2 to 0.3 kWh/t.

1.2.4 Impact crushers

In its mode of operation and design features the impact crusher differs considerably from the slow-running jaw, gyratory and roll crushers, which reduce the material by a predominantly compressive (and therefore truly "crushing") action. The

alternative terms to "crushing" and "crusher" are "breaking" and "breaker", and it would perhaps be more accurate to speak only of "impact breaker", but in practice the distinction is seldom consistently made.
In the impact crusher the feed material entering the crushing chamber encounters the impactor bars immovably mounted on the rotor and revolving with it at a circumferential velocity of 30–45 m/second. The fragments are flung against the upper breaker plate, rebound into the crushing chamber, are again subjected to the action of the impactor bars, and so on until they have been sufficiently reduced to pass through the upper gap into the space between the two breaker plates. Here the process is repeated until the material is fine enough to pass through the second gap. Besides the impact of the rock fragments with the bars and plates there is also an "autogenous" reduction effect due to the rock fragments colliding with one another (Fig. 10).

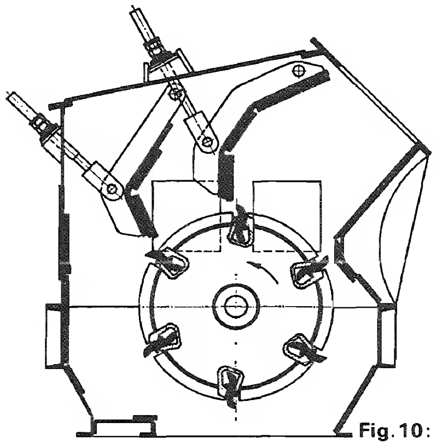
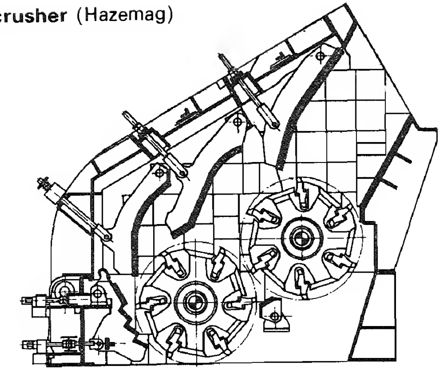


Fig. 10: Single-rotor impact crusher

The impact crusher is best suited for dealing with brittle hard to medium-hard material with natural cleavage planes. It cannot cope very well with soft, plastic and moist material.
The shape and arrangement of the breaker plates, the circumferential velocity of the rotor and the number and design of the impactor bars should be chosen with due regard to the nature of the feed material (type of rock) and the maximum feed size.
Depending on the hardness and size of the feed, coarse impact crushers reduce the material to a product size of between 150 and 200 mm and attain reduction ratios of between 6:1 and 20:1. The circumferential velocity of the rotor is a major factor: low velocity results in a coarse product; with higher velocity the size reduction

Fig. 11: Compound impact crusher (Hazemag)



energy is greater and the material is broken up into correspondingly smaller fragments, but the rate of wear on the bars and plates is of course higher (it increases proportionally to the square of the velocity). The optimum feed material size range of 0–25 mm for raw mills cannot be achieved in a single pass through the coarse impact crusher. Applying the closed-circuit principle in this case does not achieve any worthwhile improvement in reducing the product size. A more efficient method is to use a secondary crushing stage, e.g., in the form of a second impact crusher operating with higher rotor circumferential velocity.
In the compound impact crusher the two stages – primary and secondary – are combined in a single machine (Fig. 11). This is a dual-rotor crusher in which the primary rotor runs at about 35 m/second and the secondary rotor (mounted below and to one side of the primary) runs at about 45 m/second circumferential velocity. The maximum product particle size is determined by the bottom gap formed by an additional ridged comminuting anvil plate. Compound crushers can accept feed lumps up to about 1.5 m size, reducing it to a product in which 95% is smaller than 25 mm, corresponding to a reduction ratio of 60:1, achieved in a single pass.
The upper rotor is fitted with fixed impactor bars, while the lower rotor has impactor bars or movably mounted hammers, depending on the nature of the feed material and the required product fineness.

1.2.5 Hammer crushers

The hammer crusher is the most widely used machine for the primary reduction of medium-hard to hard limestone and marl in the cement industry.
The main feature is the rotor which carries a series of pivoted hammers. When the rotor is running, the centrifugal forces cause the hammers to point radially outwards. In the upper crushing chamber the feed material is subjected to a combination of impact and percussive action by the hammers and by repeated collision with the breaker plate, together with "autogenous" action by fragments

of rock colliding with one another. The finer reduction is accomplished in the gap between the hammers and the breaker plate in the single-rotor hammer crusher. The width of the product outlets between the bottom grid bars determines the maximum product particle size. As a rule, the process engineering requirement of obtaining the finest possible mill feed in a single crushing pass is fulfilled by the hammer crusher.

Hammer crushers are built as single-rotor (Fig. 12) and twin-rotor machines (Fig. 13). The rotors may consist of a series of discs mounted on a square shaft or may alternatively take the form of rollers. If hammers with forged-on individual pivot stubs are employed, the rotor discs must be axially movable for changing the hammers when they have become worn. It is, however, better to key the discs securely to the rotor shaft and to mount the hammers on continuous spindles extending the full width of the rotor. On disc-type rotors with recesses and on roller-type rotors the hammers are installed in a staggered arrangement so as to give complete coverage across the rotor. Rotors are mounted in plain or in anti-friction bearings.

The forged or cast hammers range from about 30 kg to 200 kg in weight, according to the size of the crusher. The discharge grids enclose the rotors through an angle of between 120° and 180° and are, more particularly in large crushers, axially or radially divided for convenience of handling in terms of weight and size. The forged grid bars are of triangular or trapezoidal cross-section (Fig. 14). Triangular bars form wider entry apertures to the product discharge openings and thus offer less resistance to the passage of the material, but wear away more rapidly so that the openings become too wide. This effect is less pronounced with trapezoidal bars, which moreover, for equal structural strength and equal width of the openings, have a larger open grid surface area than triangular ones.

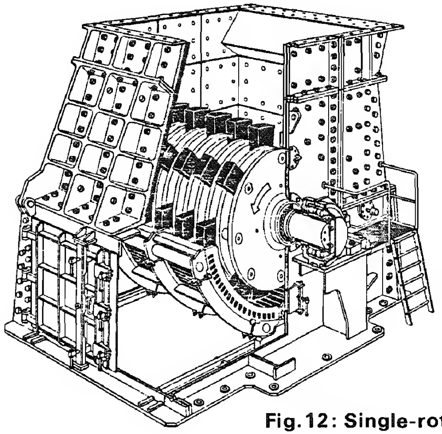


Fig. 12: Single-rotor hammer crusher (O.&K.)

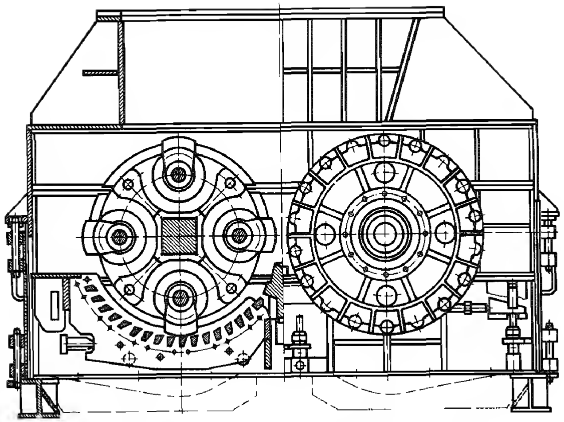


Fig. 13: Twin-rotor hammer crusher, type Titan (O.&K.)

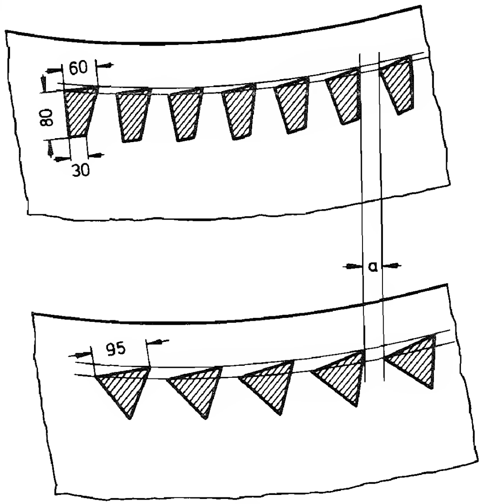


Fig. 14: Grid bar cross-sections. Effective open grid surface area F_0 for equal section modulus of bars: $F_{O_A} = 1.5\ F_{O_B}$

The grid openings of primary crushers operating as single-stage machines which supply feed for tube mills are usually 25 mm in width, thus attaining a product with only 3–5% oversize in the 25–30 mm fraction. However, widths of 40–50 mm are employed in crushers which are fed with raw materials containing plastic components and above 6–8% moisture, the greater width being necessary to avoid choking of the grids.
Single-rotor hammer crushers are built for throughputs of up to about 2000 t/hour. For example, a well known machine of this capacity has a rotor of 3300 mm width and a hammer circle diameter of 3350 mm, equipped with 112 hammers weighing 150 kg each.
The circumferential velocity of the rotors is between about 28 and 33 m/second.

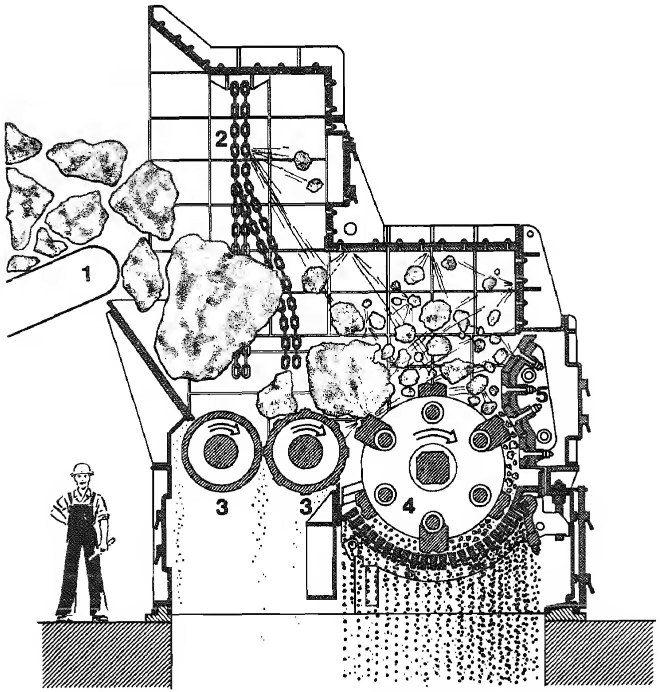


Fig. 15: Hammer crusher with feed rollers (F.L.S.). 1 feeder, 2 chain curtain, 3 feed rollers, 4 hammer rotor, 5 adjustable impact wall

The hammer crusher shown in Fig. 15 is a special form of construction, equipped with two rollers which rotate in the same direction, but at different speeds, and feed the material to the rotor equipped with freely movable hammers. Undersize particles already present in the feed are discharged through the gap between the rollers. The impact wall and bottom discharge grid, which encloses the rotor through an angle of about 120°, are adjustable in relation to the rotor, so that the wear of the hammers, breaker elements and grid bars can be compensated to some extent.
In another version of the hammer crusher there are likewise two feed rollers, but this machine has two rotors, rotating both in the same direction.

Crusher drive systems

Single-rotor and twin-rotor hammer crushers are usually driven by slip-ring motors via a V-belt transmission system. As a rule, slip resistors are provided in order to ensure flexible behaviour of the drive motor. In the event of a drop in rotation speed due to impact loading of the rotor, the motor will still develop a high torque and the V-belt drive will be less severely strained. The drive pulley, which serves as a flywheel, is overhung-mounted on the rotor shaft – even on the largest crushers hitherto built – and is fixed by means of locking sleeves or similar devices.
The motor shaft is connected to the intermediate drive shaft by means of a flexible coupling. If a squirrel-cage motor is used, a fluid clutch is additionally provided, in order to facilitate motor starting and prevent surges in the supply system.
Some twin-rotor hammer crushers are equipped with rotors directly driven through reduction gear units.
A new type of drive – direct drive with a travelling-wave (linear) motor of segmental design – has been used for single-rotor hammer crushers. In this type of motor the torque is transmitted to the rotating element, which is designed to function also as a flywheel and is mounted direct on the rotor shaft of the crusher, thus dispensing with the V-belt transmission. The travelling-wave motor has a favourable torque characteristic and takes up less space than conventional motors. Its slightly lower efficiency is hardly an important drawback, but a more serious objection is its high cost.

Auxiliary equipment

With the evolution of crushers to larger and larger throughput ratings the dimensions and weights of their wearing parts are correspondingly increasing. Removal and renewal of worn parts without the aid of suitable lifting appliances is awkward and time-consuming. The solutions adopted by some manufacturers to ease these problems will be briefly described by way of example.
Several of them have developed special auxiliary equipment to facilitate the operations of changing the wearing parts of their machines and thus substantially reduce the repair downtime periods.

Thus, hydraulic pull-out systems for extracting the breaker plates or bars are provided. Furthermore, hydraulic rams mounted on the crusher casing enable the breaker wall and certain parts of the casing to be swung open on impact crushers and hammer crushers, without the aid of other auxiliary devices. Also, various solutions for changing the bottom discharge grids of hammer crushers have been devised. Polysius, for instance, releases the bottom part of the casing and pulls out the two halves of the curved grid, which can then be lifted out by a crane. In the Miag Titan crusher the rear walls can be swung open by hydraulic rams, while the grids are connected by swivel mountings to the walls of the casing and can be moved by means of the rams for maintenance and also for adjustment while the crusher is running.
Design features on the upper part of the machine enable sections of casing which are situated beside the rotor shaft to be removed, without having to dismantle the upper casing, for taking out and refitting the rotors (Fig. 16). The twin-rotor crushers of Krupp are likewise equipped with hydraulic rams with which the rear walls can be swung open, so that the lining and rotors become accessible for

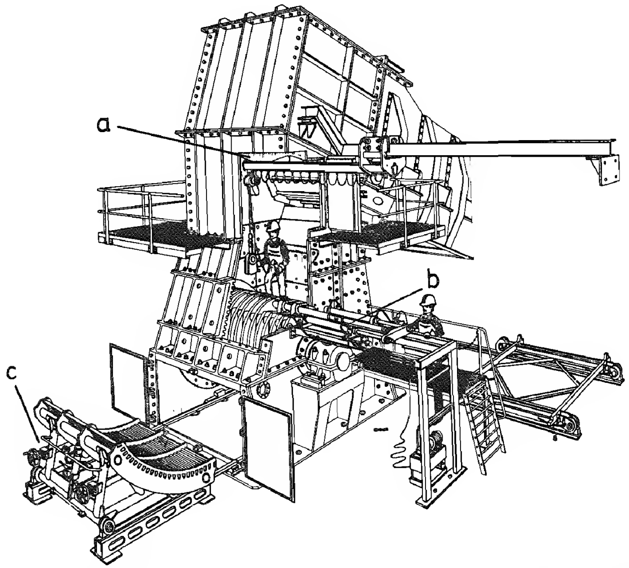


Fig. 16: Auxiliary equipment for changing worn parts (O.&K. Mammut crusher), a hammer lifting device, b hammer spindle extracting device, c discharge grid extracting device

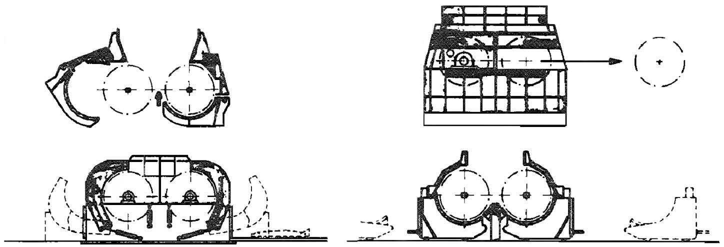


Fig. 17: Various systems for removing and refitting the discharge grids of twin-rotor crushers

inspection. Polysius has combined the grids and casing rear walls into carriages which can be moved with the aid of hydraulic rams. The Mammut (Mammoth) crusher has discharge grid carriages which travel into the crusher casing and are positioned under the grid halves to be removed (Fig. 17). After release of the lateral connections the grid is lowered hydraulically onto the carriage, so that it can then travel out of the casing.
The continuous spindles on which the hammers are mounted are extracted and refitted with hydraulic devices. An electric chain hoist can be introduced through an access door in the upper casing into the interior. In this way each hammer to be dismantled can be lifted out of the crushing chamber.
Auxiliary equipment in a wider sense comprises hammer drills – hydraulically powered, as a rule – which are installed on telescopic mountings near the feed hopper and apron conveyor and can be used to break up any oversize pieces of rock that get into the hopper. The same method is used also for dealing with such pieces that inadvertently reach the hoppers on jaw crushers or gyratory crushers.

Wear

The throughput rate and fineness of the product are affected by the state of wear of the comminuting components of the crusher. Hammers have to be reversed, resurfaced with hard steel (by welding), or entirely renewed, before their size reduction effect decreases too much. The bottom grid bars, too, must be resurfaced or renewed before they let through an unacceptably high proportion of oversize particles in the crushed product.
The hammers and grid bars are made of forged, cast or rolled steel. The choice of construction material depends on the size, hardness and abrasiveness of the crusher feed and also on the shapes that the designer adopts as being most appropriate for these parts to meet the requirements. Generally speaking, a higher factor of safety against fracture will be obtained at the expense of wear resistance.

It is advantageous to use steel having a constant tensile strength over the whole length of the hammer, with a high degree of hardness at the head and with adequate toughness and wear resistance at the pivot hole. The hardness of the metal around the hole is of major influence on the service life of the hammer spindle and is a factor that deserves careful attention in choosing the material for the hammer to be suitably compatible with that of the spindle.
If the discharge openings in the bottom grids are narrow and the feed material has a high moisture content, the striking faces of the hammers should have sharp edges. When the hammers have been worn away by an amount corresponding to about 10% weight loss, they should be resurfaced in order to preserve their comminuting effectiveness and to prevent the throughput rate of the crusher from declining. The materials of which the hammers are made should therefore also be suitably weldable, a property which is only to a limited extent compatible with the requirements of a high degree of hardness and a long service life. Austenitic manganese steels, possessing good weldability, are best suited for the purpose.
In the development of composite cast hammers (Magotteaux) with the comminuting head made of high-carbon cast steel with over 3% C and 16% Cr the possibility of resurfacing was relinquished from the outset. The head preserves its effective shape and, under appropriate conditions, the working life is more than doubled in comparison with that of the usual hammers. Lifetime is limited by the low tensile strength in the region of the pivot hole and by the restricted height of the high-carbon cast steel head, which gives rise to cavitation phenomena at the transition to the parent material.
The net rate of wear on hammers for the reduction of limestone and marl is in the range of about 0.5 to 6 g/t. Grid bars usually last at least as long, and anything up to about twice as long, as the hammers.
Considerations of economy must decide whether to use hammers of high-carbon cast steel which is unsuitable for resurfacing or instead to make use of a less resistant material which can be resurfaced. The operational availability of the crusher, wage costs and material consumption are factors to be taken into account in connection with this. The general trend is towards the use of high-strength materials offering long service life.

1.3 Crushing plants

A distinction is made between single-stage and multi-stage plants, according to whether the desired product size is attainable with just one crusher or requires two or more crushers operating in series. Each of these crushing stages may in principle be operated in open circuit (with or without preliminary screening) or in closed circuit (with screens or grizzlies as the classifying devices).
Stationary crushing plants, i.e., installed in a fixed location, are predominant in the cement industry, but for new installations, especially when large throughputs are required, mobile plants – self-propelled or easily relocatable – have become much more numerous since the early 1960s, now that the various systems for moving them from one working position to the next have proved reliable.

1.3.1 Stationary crushing plants

In the European cement industry, which uses chiefly marl and medium-hard to hard limestone as its principal raw materials, **single-stage crushing plants** equipped with hammer crushers are the commonly preferred type.
The feed hopper, feeding equipment, crusher and product removal conveyors are the main component units of the plant. The feed hopper should have a capacity equal to at least twice that of the largest vehicles supplying rock to the crusher (Fig. 18). Caking of moist and sticky feed material can be minimized by using a well designed hopper, with rounded transitions from the end walls to the side walls. If the hopper is of concrete, it should be lined with steel plates or, preferably, with steel rails, which give much better protection against wear.
Robustly constructed apron conveyors have proved most suitable for feeding. They fulfil all requirements applicable to a feeding system in order to obtain optimum utilization of the crusher: control of the handling rate within a certain range, controllability in response to the loading condition of the crusher, feed over the full working width of the crusher, ability to start under load, feed can be stopped instantly (no after-trickle of material that could choke the slowing-down or stopped crusher).
Particularly with moist feed material it is important that the apron conveyor should have the same width as the crusher rotor, so as to ensure that the rotor is fed

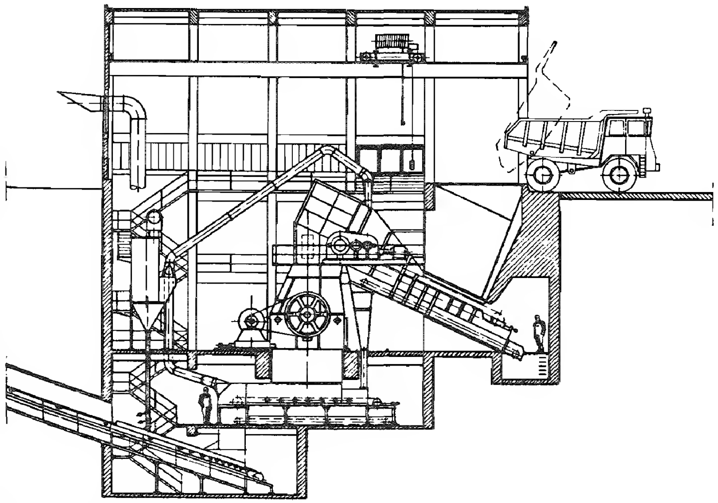


Fig. 18: Stationary crushing plant (O.&K.) **with hopper and apron feeder**

uniformly over its full width and undergoes uniformly distributed wear. Optimum utilization of the crusher is obtained by means of an infinitely variable apron conveyor drive interlocked with the crusher drive. A frequently employed method of control is to vary the apron conveyor speed in response to the rotor speed. A more suitable solution, quicker and more sensitive, is obtained by basing the control action on the power consumption of the crusher drive motor.

Rubber belt conveyors are usually employed for carrying away the product discharged from crushers equipped with bottom grids. In order to prevent caking and build-ups of adhering material, the belt should preferably be so wide as to comprise the whole width of the crusher discharge opening, so that the side walls of the connecting chute between the crusher and the belt conveyor can be made vertical.

Steel apron conveyors or chain conveyors, though mechanically more elaborate and expensive than belt conveyors, are preferable for product removal from impact crushers and from hammer crushers not equipped with bottom grids. The reason is that the material discharged from the crusher falls with greater impact force on the conveyor than when a bottom grid limits the size and impact velocity of the pieces discharged.

On the feed side of the crusher, fine material sticking to the apron conveyor and falling off it on the return run is removed by a scraper conveyor installed in a concrete trough in the foundation slab or otherwise, if accessible space is required under the apron conveyor, in a steel trough mounted directly under the latter. As an alternative solution the product removal conveyor on the discharge side of the crusher may be extended rearwards to underneath the throw-off end of the feed apron conveyor and thus catch the material falling off. In that case a separate scraper conveyor can be dispensed with.

Multi-stage crushing is employed mainly in older plants whose equipment dates from a time when high-capacity crushers with high reduction ratios were not yet available. In general, crushing in two or more stages will be applied in cases where the hardness or abrasiveness of the feed material is likely to cause considerable crusher downtime and attendant cost.

Gyratory crushers, which are used as first-stage machines when very hard and coarse feed material has to be reduced, can receive the material direct from the truck, without the interposition of a feed hopper (Fig. 19). As the preliminary crusher delivers a product in the 300–500 min size range, the second-stage crusher can function under less severe operating conditions than if the size reduction had to be performed all in one stage.

While gyratory crushers can often be employed also as second-stage machines, high-speed machines such as impact crushers or hammer crushers are more advantageous for obtaining a finer product which is suitable as feed for the grinding mills. If it is essential to feed the mills with a finely crushed product from which oversize pieces are strictly excluded, it is necessary to classify the second-stage crusher product by screening and return the oversize to the crusher for further reduction (closed-circuit operation). However, if the second stage of crushing is performed by a hammer crusher with bottom discharge grid, such classification will not be necessary. If the first stage of size reduction is performed by jaw

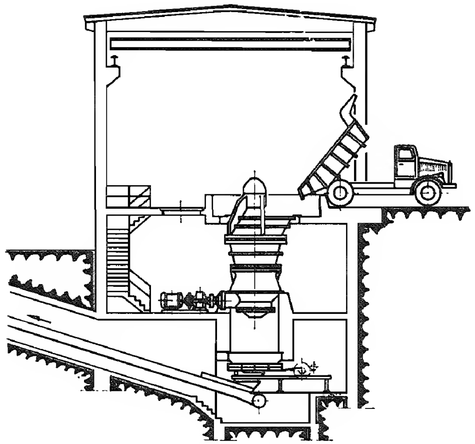


Fig. 19: Stationary crushing plant with direct feed dump trucks

crushers of gyratory crushers, which are relatively immune from overloading and can therefore be fed direct from the trucks, the product should be intermediately stockpiled to allow a uniform rate of feed to the second crushing stage. An intermediate bunker with an extracting conveyor controlled in response to the power consumption of the secondary crusher drive motor ensures that this crusher can operate under optimum conditions. From the point of view of the overall performance of the primary size reduction system it is generally more advantageous also to apply such controlled feed to the first-stage crusher – of whatever type – through a feed conveyor and hopper, in which case the second-stage crusher, designed with an appropriate safety margin of capacity, can be fed direct with the product of the first stage of crushing.

Preliminary screening

Separation of the finer particles from the raw stone before it is fed to the crusher is not standard practice in the cement industry. In exceptional cases, however, the material may first be passed over a grizzly or a reciprocating grid screen. Preliminary separation of the coarser from the finer material can serve to relieve the crusher or to improve the quality of the raw material by raising the concentration of certain desirable constituents. As a rule, it makes for better performance of the crusher, too.

Relieving the crusher

The decision as to applying preliminary screening of the raw stone is governed by the proportion of fine particles in it, the physical properties of those particles, and the technical design features of the crusher employed. The possibility of thus relieving the crusher may, in new plant design, result in deciding to use a smaller machine than would otherwise be required. Also, the subsequent installation of preliminary screening in an existing plant can bring about an improvement in crusher performance – higher throughput rate – without involving major capital expenditure (Fig. 20).
Removal of the fines from the crusher feed reduces wear of the crushing elements, besides cutting down the hazard of clogging and caking in the crusher.
Comparisons of capital expenditure and operating experience show that the additional installation of mechanical equipment for preliminary screening is profitable only if about one-third of the material flow supplied to the crusher can be separated in this way. For a given feed material a crusher can be relieved to a greater extent according as the reduction ratio that it can attain is lower. This is particularly true of jaw, gyratory and roll crushers as compared with hammer and impact

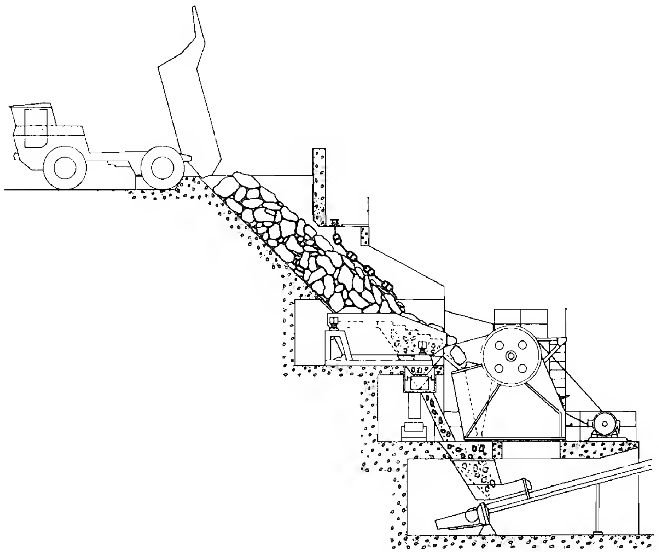


Fig. 20: Static primary crushing plant with preliminary screening on stepped (multi-stage) grizzlies (Babbitless)

crushers. The preliminary removal of fine, sticky and moist material may be advantageous in reducing the risk of clogging and diminished performance, particularly with jaw and gyratory crushers, but also with hammer crushers.
In general, however, preliminary screening offers no advantage when primary size reduction is done in a single stage in hammer crushers, which attain high reduction ratios and can deliver a product below 25 mm particle size, commonly considered to be the maximum acceptable as ball mill feed. As a rule, the raw stone seldom contains more than 15–20% of fine particles, so that their preliminary removal from the crusher feed is hardly worth-while. The separation of moist sticky material below about 25 mm size in the crusher feed can moreover be problematical and can only be accomplished with poor efficiency.
The preliminary screening devices used before primary crushers are various types of stationary grizzlies or moving grid-type screens (with bars or with rollers, either round or elliptical), reciprocating separators, vibrating grates or heavy eccentric-weight-driven shaking screens (Fig. 21). A relatively recent development is the Mogensen sizer, which is especially suitable for the separation of moist fine material that is difficult to remove by screening from the crusher feed (Fig. 22). This

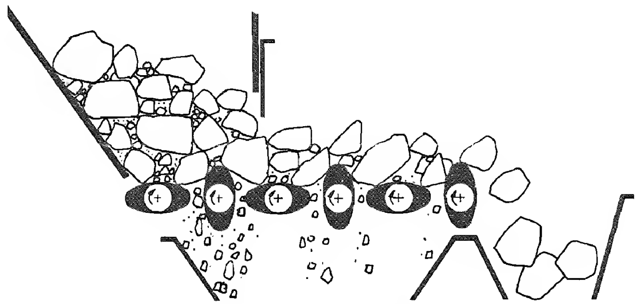


Fig. 21: Grizzly with elliptical rotating rollers

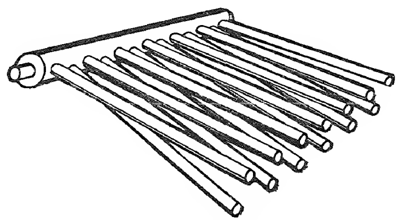


Fig. 22: Mogensen sizer (illustration of its principle)

machine comprises a number of round steel bars individually attached to a transverse tubular member. The bars are not all in the same plane, but are set in a staggered arrangement. This ensures that the effective size of the apertures increases in the direction of flow. The bars, up to 2 m in length, oscillate in response to the weight of the material moving over them; this oscillatory motion helps to prevent choking. The range of performance of Mogensen sizers is stated to be characterized by cut sizes from 300 mm down to 25 mm. If sharpness of separation is of major importance, two or more sizers may be operated in series.
The design principle is simple, no drive power is required, and renewal of worn bars can be accomplished with relatively little effort.

Intermediate screening

With multi-stage crushing, removal of the fine particles from the first-stage product can result in notably relieving the second stage. Screening applied after the final crushing stage, i.e., directly before the grinding mill, is advantageous if the crushers yield a product with a high proportion of oversize which is liable to cause trouble in mill operation.

Feeding of two components

Moist and plastic clays are difficult to comminute with their natural pit moisture content. They can be crushed, without simultaneous drying, in roll crushers, operated multi-stage because of their low individual reduction ratios. There may be difficulties not only with comminution, but also with storage, reclaiming and feeding to the grinding mill if such plastic material is handled alone. Combined crushing of limestone and clay in the limestone crusher is more favourable. For this purpose the two materials, in their correct quantitative proportions, are tipped into the feed hopper. This procedure does not, however, achieve sufficiently homogeneous blending of the materials.
A solution which is both favourable from the process engineering standpoint and relatively simple in terms of mechanical equipment consists in feeding the two components, at controlled rates, from separate feed hoppers, each delivering its material by its own feeder (Fig. 23). Thus, by means of two apron feeders with speed control, the crusher can be fed with a correctly proportioned mixture of raw materials which conforms quite closely to the specified chemical composition. The limestone-clay mixture can usually be handled without difficulty by the hammer crusher even if the clay has very unfavourable physical properties.
Coarse hard limestone as the main component of the mixture performs a cleaning action in the crusher and facilitates the combined reduction of this material with the plastic clay tending to clog the machine. In proportioning the two components the clay is deposited onto the limestone. The speeds of the two apron feeders are so interadjusted that the desired mixture is supplied to the crusher. The two feeder drives are coupled together in such a way that any change in the speed of the main apron feeder in response to the power consumption of the crusher drive motor is immediately followed by a corresponding change in the speed of the secondary apron feeder (which handles the clay component) so as to ensure that the predetermined mix proportions are maintained.

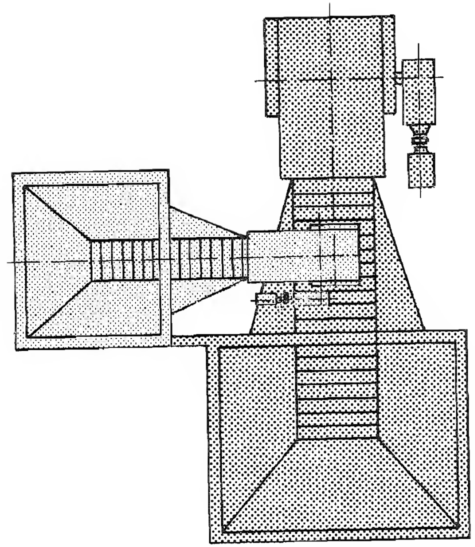


Fig. 23: Simultaneous feeding and crushing of two components with two apron feeders (O.&K.)

Proper design of the clay hopper is very important. Its walls should be as steep as possible, and preferably be plastic-lined, in order to prevent the clay from sticking to it. The handling appliance – apron conveyor, chain conveyor or belt conveyor – should not be too narrowly dimensioned, even if only quite small quantities of clay have to be handled, because otherwise arching of the material between the side guide plates of the conveyor is liable to occur, giving rise to trouble with feeding the clay.

Protection against foreign bodies

In the quarrying and loading of raw materials it inevitably occurs that metallic foreign bodies – excavator bucket teeth, broken drill rods, drill bits, pieces of rail, chains, etc. – turn up in the feed material supplied to the primary crusher. If the crusher is fed direct by excavators or dump trucks, there is no opportunity of intercepting and removing such pieces of metal. Nor is it possible to remove them from the feed material flow: the size of the rock fragments and the very considerable depth of the moving material (sometimes more than 1 m) rule out the use of tramp iron separators.

In primary size reduction, more particularly with big single-stage coarse crushers, operational reliability is best achieved by very heavy and robust design of these machines, equipped with mechanically or hydraulically operated overload protection devices to prevent damage being caused by foreign bodies that cannot be crushed. On the other hand, secondary crushers which are fed with pre-crushed rock below about 300 mm particle size can be effectively protected by magnetic separators by metal detectors.

Overload protection

The toggle plates in jaw crushers may be designed as "predetermined fracturing components", i.e., designed to fail first in the event of overloading of the machine, thus forestalling more serious damage to other parts. Hydraulic overload protection systems are more expensive, but they avoid having to stop the plant for replacement of a fractured toggle plate. The stationary crushing plate can swivel about a top pivot, while lower down it is held in its normal working position by hydraulic rams. When a large piece of uncrushable material enters the crushing chamber, an overpressure develops in the hydraulic system, causing the crushing plate to swing aside. As a result, the foreign body drops through the discharge opening, the rams move the crushing plate back to the working position, and the feeder – which was automatically stopped when the hydraulic overpressure developed – is restarted. With this protective system the standstill periods due to overloads are substantially shortened.
The same principle of hydraulically controlled "give" has been applied to gyratory crushers: if a large piece of metal becomes lodged in the crushing chamber, the discharge opening widens to let it pass. A similar purpose is served by the movable rolls, held in the working position by springs or hydraulically, on double-roll crushers. In impact crushers the breaker elements are similarly designed to move aside and thus prevent overload damage to the impactor bars or plates.
Crushers with bottom discharge grids, especially twin-rotor machines, which pull the feed material and any foreign bodies it contains into the crushing chamber, are more seriously at risk. Single-rotor hammer crushers are less prone to overload hazard if – as, for example, in the Mammut crusher (Fig. 16) – any pieces of metal entering the crusher are hurled against the breaker plate and rebound back out of the crushing chamber onto the feed conveyor. However, the feeding system will then have to be stopped and the metal removed by hand. An advantageous feature is the use of hammers which can rotate freely through 360° on their pivots and can thus swing aside if they encounter uncrushable foreign bodies.
As a rule, foreign bodies can be more effectively removed from the material after it has been pre-crushed (first-stage crushing). Magnetic separators and metal detectors are used for the purpose and help more particularly to protect the high-speed second-stage crushers.
Drum-type electromagnetic separators comprise a stationary set of magnets surrounded by a horizontally mounted rotating drum or cylinder made of a non-magnetic material. The crushed material is passed over the drum, and any tramp iron contained in it remains clinging to the drum and is carried round to the underside thereof, where there is no magnetic field, so that the pieces of iron fall

off. Powerful electromagnet systems are necessary for dealing with coarse material moving in a stream of great depth. For effective action of the separator it is essential to distribute the material evenly across the full width of the drum. Drum separators equipped with **permanent magnets** may be used for iron removal from fine-grained material of limited depth on the conveyor.
Electromagnetic belt pulleys, used at the discharge ends of rubber belt conveyors, are equipped with a rotating set of magnets acting around the full circumference of the pulley. Pieces of iron are carried round to the underside and fall off the return run of the belt once they have moved out the magnetic field. Magnetic pulleys are available for belt conveyors of all the normally employed widths and speeds (and for the depths of material which are determined by these operating parameters).
Suspended magnets are installed over belt conveyors, chutes or ducts and lift the tramp iron out of the flow of material. From time to time the magnet is swung aside, and the excitation current switched off, to allow cleaning of the magnet. For dealing with material containing a substantial amount of tramp iron, **belt-type suspended magnetic separators** (Fig. 24) may be used. A device of this kind is equipped with a continuous rubber belt which carries the pieces of iron out of the magnetic field, so that the magnet pole face itself remains clear. For reasons of space such separators are usually mounted transversely to the direction of flow of the material on the conveyor. Favourable mounting positions are the points of feed onto, or discharge from, the conveyor, because at these points the material is loosened up and the extraction of tramp iron thus made easier.
For all types of magnetic separator the rule is that any equipment and parts within range of the magnetic field should be made of non-magnetic materials, otherwise

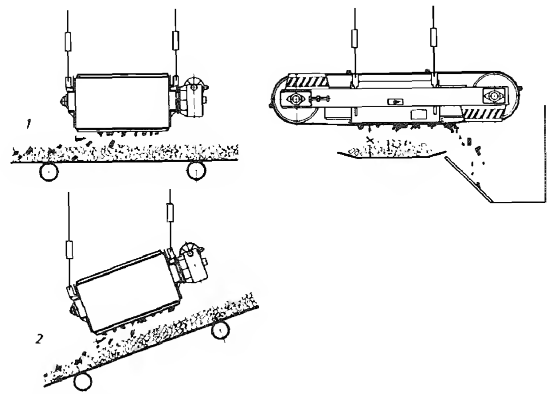


Fig. 24: Belt-type suspended magnetic separator, mounted transversely over a horizontal (1) or an inclined ascending belt conveyor (2) (Steinert)

these would be magnetized and undesirably attract iron or steel objects. The lateral clearance from tramp a iron magnet should be about 0.3 times the width of the magnet. Under the magnet a clear headroom equal to 0.7 times its width should be provided.

Metal detectors are used for revealing the presence of tramp metal which is not magnetically responsive. The equipment generally comprises one or two detecting coils installed over and/or under the belt conveyor or enclosing it. The presence of a piece of metal in the otherwise constant magnetic field of a coil causes an electric pulse which can be utilized for switching off the conveyor or causing a certain length of the layer of material on the belt, containing the metal, to be diverted from the main conveying path. Obviously, there should be no moving metal parts in the vicinity of the detecting coil. Static metal parts do not disturb the detection, but are liable to weaken its sensitivity.

Hygroscopic materials which, when moist, become electrically conductive may cause false alarms due to variations in moisture content (and therefore in conductivity) on passing the metal detector. The most reliable protection against tramp metal is provided by the combination: metal detector – magnetic drum – metal detector (Fig. 25). In this arrangement the first metal detector operates the switch-on/switch-off of the drum separator, whose magnetic field therefore is activated only when metal is detected on the conveyor. Any non-magnetic metal that passes the drum will produce a response from the second metal detector.

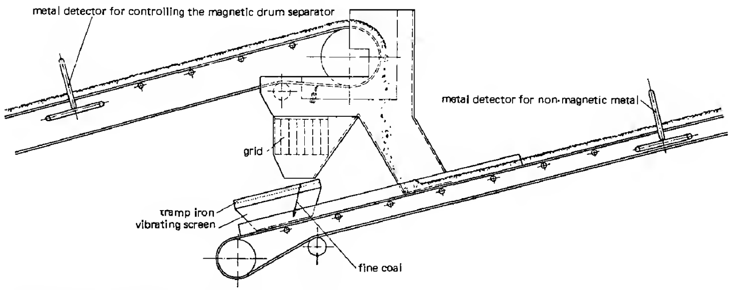


Fig. 25: Protection against foreign bodies by a combination of metal detectors and magnetic drum separator

Heating of crushers

The pre-drying of raw materials prior to primary size reduction is employed only in exceptional circumstances. Elaborate arrangements to prevent "false" air in-leakage into the heated crushers are required in such cases. As a rule, no utilizable waste heat is available at primary crushing plants which have to deal with raw materials with a high natural moisture content, especially as such plants are often at some considerable distance from the actual cement works. Besides, the low specific surface of the coarse material and its short time of sojourn in the crusher make effective drying impracticable.

The only heating systems applied to primary crushers are intended, not for drying the material, but preventing the caking of moist sticky materials which might otherwise cause clogging.

Heating the bottom plate of the feed chute, the side walls and the breaker plates in impact crushers to surface temperatures of 180° – 200° C is done with the aid of heat transfer oil circulated at approximately 300°C through a system of pipes. Heat input ratings are in the region of 20000 kcal per hour and per square metre of heated surface.

Indirect heating of certain areas of the inlet and outlet casing and crushing chamber where moist material tends to adhere has been tried out in some hammer crushers. These critical surfaces are heated with externally applied electric heating elements, with the results that caked material spalls off. Insulated hoods protect these radiant heaters and improve the efficiency of this simple and relatively inexpensive heating system, which requires little maintenance.

Determining the crusher capacity

The nominal capacity, or rating, of the crusher will be governed by the required raw material throughput and the possible working time of the crushing plant. The quarrying operations, of which the crushing plant usually forms part, are in most cases conducted on a single-shift basis with five or six working days per week. For an 8-hour shift the effective crushing time per shift can be put at 7 or at most 7.5 hours. The crusher should therefore, in an effective time of 35 to 45 hours, be able to produce sufficient raw material to feed the kiln plant for a whole (7-day) week.

The requisite crusher throughput capacity can be calculated from the following formula:

$$D_{crusher} = \frac{D_{kiln} \times v_{R/C} \times t_{kiln}}{24 \times t_{crusher} \times (1 - f/100)}$$

where:

			example:
$D_{crusher}$	capacity of crushing plant	(t/hour)	
D_{kiln}	capacity of kiln plant	(t/day)	3000
$v_{R/C}$	raw material/clinker ratio	(kg/kg)	1.6
f	natural moisture content of raw material	(%)	4
t_{kiln}	working time of kiln per week	(hours)	168
$t_{crusher}$	working time of crusher per week	(hours)	35

For the example values listed above the requisite crusher capacity is thus:

$$D_{crusher} = \frac{3000 \times 1.6 \times 168}{24 \times 35 \times 0.96} = 1000 \text{ (t/hour)}.$$

If the crusher is designed for single-shift working, it will have sufficient capacity even if the kiln plant capacity is subsequently doubled: in that case the quarry and crusher will have to work double shifts, leaving the week-ends available for repairs and maintenance.

1.3.2 Mobile crushing plants

Because of the coarse grading of the fragmented rock pile produced by blasting in the quarry, this material cannot, as a rule, be directly handled by belt conveyors. In order to be able to use belt conveying – generally less expensive than long-distance road haulage – from a point as close to the quarry face as possible, the rock pile will at least have to be crushed to "belt conveyable" size, which generally means that it should not contain pieces above about 200–400 mm. The need for crushing at the quarry face and for moving the crusher along with the site of quarrying operations has led to the development of various mobile installations in capacities ranging up to the highest throughputs required.

Depending on the method of moving the crushing plant from one working position to another, a distinction can be made between truly mobile (self-propelled) plants and semi-mobile ones (not self-propelled).

Mobile plants in the more specific sense of the term have their own integral travelling machinery, enabling them to proceed from one location to the next unaided.

Wheel-mounted (rubber-tyred) crushers are employed in cases where they have to travel over relatively long distances and have to be highly manoeuvrable. Such machines can move at speeds of up to about 6 km/hour. Under suitable conditions the running resistance of the tyred wheels is relatively low, so that drive power requirements are correspondingly modest. When the crusher is in operation, the wheels are relieved of load, either by being lifted off the ground to that the crusher is directly supported or by the lowering of strut legs producing the same effect. A drawback associated with wheel-mounted crushers is the high bearing pressure exerted on the ground (4.5–9 kg/cm^2). They can travel on gradients of up to about 1 in 10. On heavy plants, hydraulic axle load adjustment compensates for the effects of irregularities on rough ground. Sprung wheel suspension systems serving the same purpose are used on smaller and lighter ones (Fig. 26a).

Crawler-mounted mobile crushers can likewise ascend 1 in 10 gradients and travel over ground which need only be roughly cleared of obstacles. Besides, the bearing pressure is low (1–1.5 kg/cm^2). Travel speeds are between 5 and 8 m/minute. The crawler tracks are not relieved of load when the crusher is in operation and they are therefore subjected to severer service conditions than other travel systems (Fig. 26b).

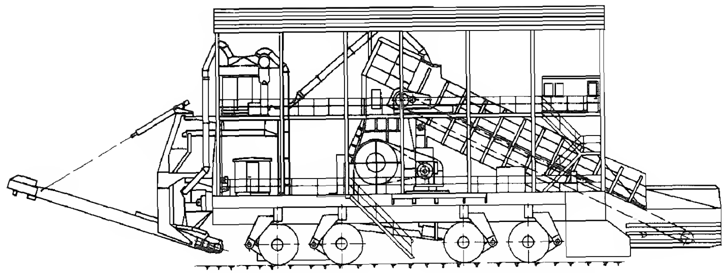


Fig. 26a: Wheel-mounted (rubber-tyred) mobile crushing plant (O. & K.)

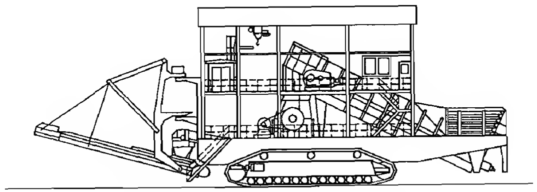


Fig. 26b: Crawler-mounted mobile crushing plant

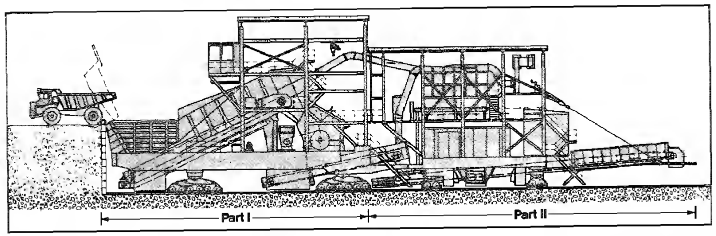


Fig. 26c: Rail-mounted mobile crushing plant comprising two sections (O. & K.)
Section I: feed hopper, apron conveyor, crusher, product conveyor
Section II: belt conveyor, dust collector, power supply system

Rail-mounted mobile crushers can suitably be used in cases where the direction of travel is well defined in advance (Fig. 26c). Thanks to the low rolling resistance, drive power requirements for travel are low, and wear on the travel machinery is light. Against this there is the disadvantage that gradients of only about 1 in 40 can be overcome, while maneouvrability is limited and moving the plant to a fresh

working location requires preparation of the quarry floor and, of course, track-laying. The ground should have a fairly high bearing capacity. As a rule, no load-relief of the travel wheels during crusher operation is provided.
In terms of cost the most favourable travel system is the hydraulically powered **walking mechanism**, usually comprising a walking pad with three lifting rams with which the whole pontoon-like platform with the crusher and other equipment can be lifted. Horizontal hydraulic walking rams installed between the platform and the walking pad enable the plant to be moved or turned in any direction (Fig. 26d). Speeds of about 0.7 to 1.5 m/minute are commonly adopted for walking crushers. While the crusher is in operation the walking pad is kept raised off the ground, the whole plant then being supported on strut legs.
Semi-mobile crushing plants are wheel-mounted (on rails or on rubber tyres) and are moved to fresh working locations by towing or pushing, i.e., they are not self-propelled. In recent years an alternative system has been to use special rubber-tyred or crawler-tracked **lifting vehicles** which bodily convey the whole plant to a fresh position. The advantage is that one and the same lifting vehicle can serve the needs of several plants and that, when not in use, the vehicle can be stored under protection from the weather and other adverse influences (Fig. 26e).
Besides mobile and semi-mobile crushing plants there are what can best be described as relocatable ones, being **skid-mounted**, so that moving them requires powerful tractors. Even so, such plants are restricted to relatively low service weights and low throughput capacities.
Which system of moving the crusher should be chosen will depend to a great extent on the technical features of the quarrying operations and on the condition (evenness, roughness) of the quarry floor.
As with stationary crushers, mobile crushers located close to the working face in the quarry can be fed directly by loading shovels or by dump trucks if they are, for example, gyratory crushers which are substantially unaffected by irregular loading. As the crusher will generally be standing on the quarry floor, the vehicles delivering the fragmented rock to it should be able to travel up a ramp to the requisite dumping height or the crusher may alternatively be fed from a higher floor level (or bench) than that on which the crusher is standing. Direct feeding of a crushing plant without the interposition of a haulage vehicle was practised in a West German cement works quarry in the 1960s. In that system the rock pile obtained by blasting was fed, with the aid of a scraper, via an inclined plane to the gyratory crusher. The crushing plant was equipped with a hydraulic walking system.
Optimum utilization of the crushing plant – whatever the type of crusher used for reducing the coarsely fragmented rock pile – can be obtained only by feeding it as uniformly as possible. As in the case of a static plant, the mobile crusher must be fed at a steady rate via a feed hopper. For direct loading by excavators or loaders the hopper is restricted to a height that enables these machines to discharge into it, so that its capacity is correspondingly limited. If larger hoppers are used, it will be necessary to build earth ramps to them or otherwise to use relocatable steel ramp structures.
If the quarrying operations require moving the crusher to a fresh position only at infrequent intervals, it will be advantageous to feed it by using heavy dump trucks

operating from a haulage level at the appropriate height above the floor on which the crusher is standing. In that case it is advantageous to build a suitably paved ramp that can safely and reliably be used by the vehicles. With such arrangements the hopper capacity can be as large as that for a stationary crushing plant.

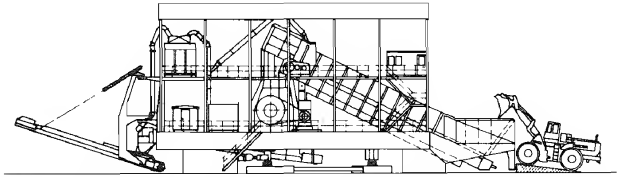


Fig. 26d: Mobile crushing plant with hydraulic walking mechanism (O. & K.)

Feed material	limestone
max. feed size	1900 mm × 1200 mm × 1000 mm
product fineness	98% < 25 mm
throughput	1000 t/hour
Type of crusher	single-rotor hammer crusher with discharge grate
Feeding system	
feed hopper	30 m³ capacity
apron conveyor	
width	2500 mm
length	22 m
drive	infinitely variable, controlled in response to crusher drive load
Product handling	
extractor belt	rubber belt conveyor (flat)
transfer belt	rubber belt conveyor (troughed), slewable through 120°
Travel system	hydraulic walking mechanism
walking speed	0.7 m/minute
max. gradient	1 in 10 max.
specific ground pressure	1.5 kg/cm²
Dimensions	
length overall	52.5 m
width	10.6 m
height	16.0 m
Weight	920 tonnes

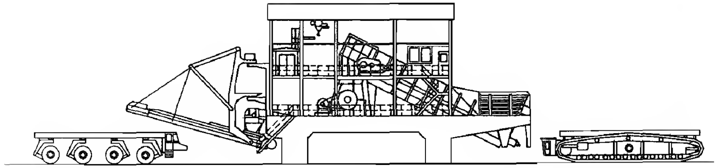
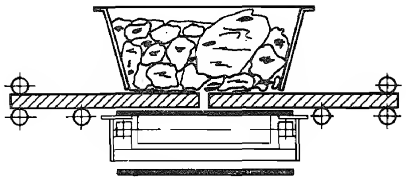
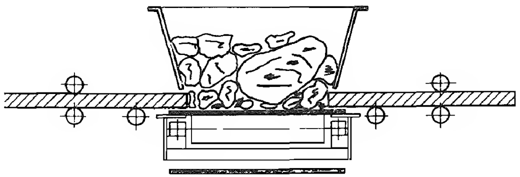


Fig. 26e: Crushing plant moved with the aid of lifting vehicles

For plants comprising a feed hopper, feeder and crusher the type of conveyor most commonly employed is the robust apron conveyor. For feeding the rock to mobile crushers, however, heavy-duty belt conveyors are occasionally used, these having the advantage of lower weight than apron conveyors, so that the overall weight of the mobile equipment is correspondingly less. To reduce the severity of the service conditions to which the belt is subjected, a special belt loading hopper with an automatically opening and closing bottom, functioning in the manner of a slide gate, may be employed (Fig. 27). Loading this hopper with rock is done with its bottom closed. With the belt conveyor temporarily stopped, the bottom gate opens, allowing the rock to fall gently onto the belt, which is then restarted and delivers the rock to the crusher. Continuous feeding of the crusher is not achieved, however.



(a) gate closed



(b) gate open

Fig. 27: Belt conveyor feed hopper with bottom slide gate (Esch)

As with static systems, the conveying equipment for removing the crushed stone from mobile plants may be belt, apron or chain conveyors. As a rule these handling appliances deliver the stone to belt conveyors which are adjustable for height, can slew through an angle of about 120° and are connected to the frame of the crusher supporting platform. These belt conveyors in turn discharge the stone onto a mobile intermediate conveyor or direct onto the extendable or retractable end of the main belt conveyor that carries the material out of the quarry.
Again as with static crushing plants, the choice of crusher will be governed by the properties of the raw material. Single-stage size reduction of the rock pile to suitable mill feed size – i. e., to 25–80 maximum particle size, depending on the mill system – is the preferred technique. Any other procedure such as multi-stage reduction, preliminary screening before the crusher or closed-circuit operation must always involve expensive additions to the mechanical equipment as compared with single-stage crushing.
Very large crushing plants are subdivided into two or more plant sections. Thus, the feed hopper and feeding equipment form a structural unit. The crusher and product discharge conveyor form another unit. These two units are separately moved from one working location to the next. Mobile lifting units, mounted on crawler tracks or on wheels, can suitably be used as the vehicular base for giving mobility to such systems.
The technical data for a single-stage crushing plant for a throughput of 1000 t/hour, equipped with a hydraulically powered walking system, give some guidance on the mechanical sophistication, the dimensions and the weights of a modern mobile installation (see Fig. 26d).

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2 Size classification

In the context of crushing and grinding the term classification means the separating or dividing of particulate bulk materials consisting of a mixture of different particle sizes into two or more size ranges or fractions. In general, separation may be effected on the basis of volume, i. e., the geometric dimensions of the particles, or on the basis of mass, i. e., differences in material properties.

Separation according to particle size is done by screening or sieving. Inertia forces are utilized in cyclone separators and in various more sophisticated devices collectively called air separators or classifiers.

2.1 Screening

In the cement industry, particle size classification by screening as part of the production operations is of less importance than, say, in the lime industry or in coal and ore preparation. Indeed, true classification procedures in the primary size reduction stage do not occur in cement manufacture, since the aim of the crushing treatment is to produce a feed material suitable for grinding to a fine powder, not the production of size fractions as required for crushed stone used in road construction, concrete production, etc.

For particle size separation in the finely pulverized products of the grinding processes in the cement industry – raw meal and cement – there is no economic possibility of classifying large quantities by screening or sieving, which has to rely solely on gravity. On the other hand, screening or sieving is of importance as a test procedure for assessing the effectiveness of crushing and grinding treatments, more particularly by determining the particle size distribution or the percentages retained or passing certain screen or sieve sizes and thereby monitoring the granulometric composition of intermediate and final products.

In crushing plants, screens with surfaces comprising usually square apertures (formed by a series of wires extending in two directions) or slots (formed by parallel bars; these are known as grizzlies) are used for the following purposes:

(1) relieving the operating conditions of primary crushers by preliminary separation of fine particles from the feed;
(2) removal of unsuitable constituents such as sand or loam, in order to enrich or concentrate the lime component;
(3) separation of the product of a primary crusher into fine and coarse fractions; the latter may be returned to the crusher for further size reduction or be fed to a secondary crushing stage.

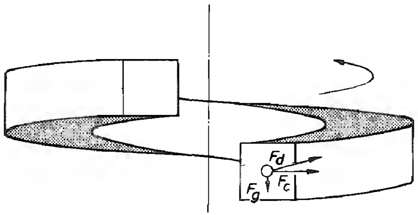
In general, obtaining a sufficiently finely comminuted product, i. e., with particles not exceeding a specified upper size limit, may be important for the protection of subsequent size reduction machinery (secondary crushers, grinding mills) or for ensuring favourable operating conditions in subsequent processing stages (grinding, pre-blending).

In the further stages of cement manufacture, screening is used for the following purposes:

(4) screens are interposed into the product flow from clinker plants in order to remove any oversize clinker particles or fragments of fractured grinding media;
(5) screens installed before packing plants and bulk cement dispatch facilities serve to protect the machines and equipment from trouble that could be caused by foreign bodies or lumps of material;
(6) in clinker dispatch installations, screens are used for the removal of fine particles caused by abrasion or shattering, thus reducing dust nuisance in the handling of the material.

2.2 Classification associated with dry grinding processes

Types and operating principles:
For very fine size reduction using closed-circuit operation it is necessary to separate the particles fine enough to qualify as "finished product" from the coarser particles (oversize) in the product discharged from the grinding mill. The requirements that the separator, or classifier, must fulfil are; good selectivity (sharpness of separation) to enable economical grinding plant operation, and highest possible uniformity of the granulometric composition of the finished product.
In the dry system of closed-circuit grinding the separation is effected in various types of air-swept devices called air separators or air classifiers and often comprising power-driven rotating elements (in which case they are called mechanical air separators). In the wet system, the classifying devices are screens or hydrocyclones.
Various types of air separator employed in connection with cement manufacture will now be described. They all function on the same principles. A particle in a rotating current of air is subjected to the interaction of three sets of forces: the force exerted by the air (proportional to the square of the mean particle diameter), the force of gravity, and the centrifugal force (the two last-mentioned forces are governed not by the size, but by the mass, of the particle). If the effective force exerted on the particle by the air exceeds the resultant of gravity and centrifugal force, the particle will remain airborne and be carried along with the air. If the force of gravity prevails, the particle will sink, and if the centrifugal forces prevails over the other forces acting on the particle, the latter is flung outwards against the wall of the separator, where its motion is arrested so that it is then precipitated as in an ordinary cyclone separator (Fig. 28).
Although the separators used in the cement industry are broadly similar in principle, they differ considerably from one another in matters of design and range of application. The differences consist mainly in the method of introducing the



F_d force exerted by the air (air drag)
F_c centrifugal force
F_g force of gravity

Fig. 28: Forces acting on a material particle in a rotating current of air

material and the separating air, the magnitude of the centrifugal acceleration, and the method of separating the finished product (the fine particles) from the air stream. In some air separators the material can moreover be given a drying or a cooling treatment.

2.2.1 Static air separator (Fig. 29)

The static air separator or classifier is so called because it has no moving mechanical parts. It is used chiefly in conjunction with air-swept grinding plants (operating with tube mills or roller mills). The material to be classified is carried along in a stream of air from the mill and enters the separator from below. It flows between the conical outer casing and the inner separating cone. As a result of the

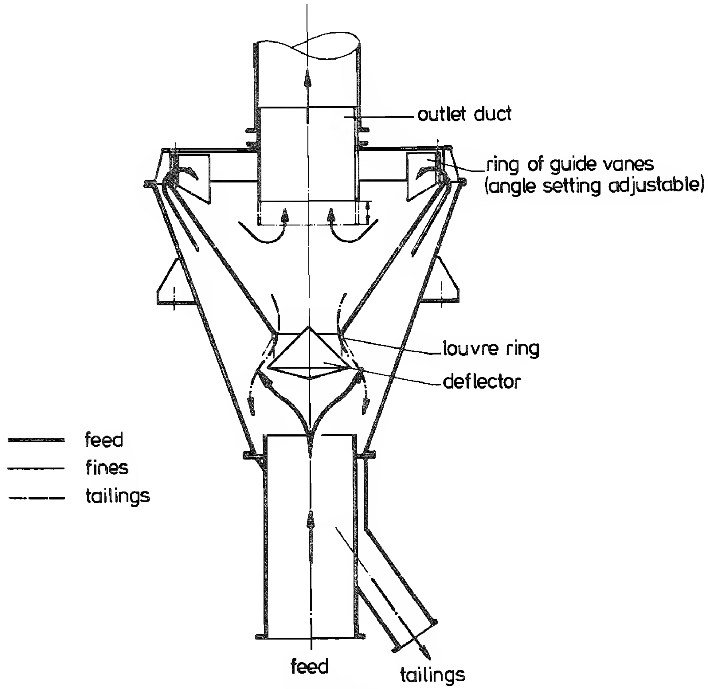


Fig. 29: Static air separator (schematic)

increasing cross-section the air flow velocity is reduced here, and coarse particles are precipitated. At the same time, the tangential admission of the air brings about a rotational motion in this outer separating chamber, so that a certain amount of centrifugal precipitation also occurs in it. The material collected here is discharged through the tailings (oversize particles) outlet of the separator.
In the upper part of the separator the material-laden air enters the inner cone through a ring of adjustable guide vanes. The material particles will be subjected to a centrifugal acceleration whose magnitude depends on the setting of the vanes. Just as in a cyclone separator, the air carrying the material spirals downwards in the inner cone and is accelerated in doing so. The result of the force of gravity and the centrifugal force thus prevails over the force that the air exerts on the larger and heavier particles, which are flung against the wall of the cone, where they lose their velocity and slide down the wall into the tailings outlet of the separator. The tailings are returned to the mill for further grinding. On the other hand, the smaller particles (the fines) remain entrained in the air, which carries them upwards in its spiralling motion and out of the separator. This discharged air laden with fines (the finished product of the grinding process) is passed into a product collector – usually a cyclone or a filter – in which these particles are finally separated from the air.

Control possibilities:
The separation characteristic of the static air separator can be varied in several ways:

– by varying the air flow rate and therefore the velocity of the air, which in turn alters the force exerted by the air on the particles and indirectly also the centrifugal force to which they are subjected.
– by adjusting the deflector over the bottom inlet duct through which the material-laden air enters the separator

In some separators of this general type the position of the deflector can be adjusted in the vertical direction. Reduction of the distance between deflector and the mouth of the inlet duct causes intensified acceleration and change of direction of the air stream. The material particles impinge on the wall of the casing and fall into the tailings outlet. This classification by deflection and impingement is rather unselective, and for this reason small distances between inlet duct and deflector are used mainly in cases where the static separator has to act as a dust precipitator, e. g., as a pre-collector, and not for the sharp separation of particle sizes. Besides, this classsification involves excessive pressure loss in the system. In more sophisticated forms of construction the deflector, which in its simplest form may be a mere baffle plate, is given a streamlined conical shape and may be fitted with attached guide vanes by means of which a laminar spiral flow pattern of the air entering the separator can be obtained. With this arrangement the precipitation of the particles in the outer chamber is accomplished chiefly by the cyclone wall effect already mentioned.

– by adjusting the top outlet duct

As in ordinary cyclones, the cut size – the particle size at which separation between fines and oversize is effected – can be varied by vertical adjustment of the

air outlet duct at the top of the separator. For a constant air flow rate, an increase in length of this duct will, within limits, shift the cut size so as to give a finer product, and vice versa. Although most static separators have an adjustable top outlet duct, this does not constitute a suitable method of routine product fineness control, but serves as a means of basic adjustment to suit the given operating conditions.

2.2.2 Bladed rotor separator

The characteristic feature of this type of air separator is a rotor comprising a set of blades in a conically tapered arrangement and rotating on a vertical shaft in a casing of truncated-conical shape (Figs. 30a and 30b). The material-laden stream of air is admitted from below and is distributed sideways by deflection at the underside of the assembly. The rotating blades accelerate the rotational flow of the air which already has a spiral motion as it enters the separator casing. The rotational and accelerational effects are intensified by the upward narrowing of the space between the rotor and the outer casing. The air is drawn inwards by suction through the gaps between the rotor blades. The heavier particles, i. e., those for which the resultant of gravity and centrifugal force prevails over the force exerted on them by the air stream, are flung outwards against the wall of the casing and then fall back into the mill (Fig. 30a) or into the tailings outlet (Fig. 30b). The fine particles are carried out of the separator with the air and are precipitated from it in cyclones or in filters.

Control possibilities:
For constant air flow rate the performance of the separator can be modified by varying the rotor speed. Because of the effect on the performance of the grinding mill, variation of air flow rate is possible only within limits.

a material-laden air from grinding chamber and tailings return
b fines discharged from separator (product)
c rotor with blades,
d variable drive

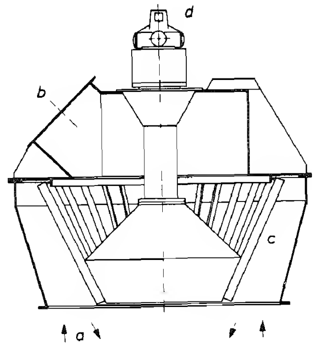
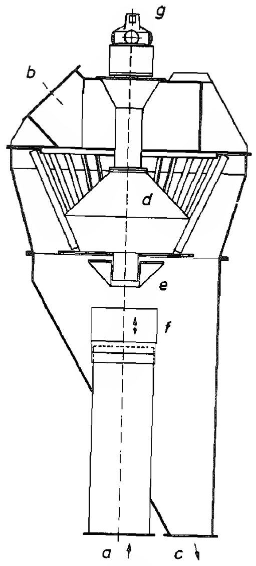


Fig. 30a: Bladed rotor separator, with drive, as mounted over roller mills
(Loesche GmbH)



a feed material in gas stream,
b fines discharged from separator (product)
c tailings discharged from separator (coarse particles)
d rotor with blades
e adjustable cone
f adjustable air control ring
g variable drive

Fig. 30b: Bladed rotor separator, with drive, as an independent classifying unit fed through a riser duct (Loesche GmbH)

Range of application:
Because of its ability to accept high air throughputs the bladed rotor separator is used mainly in conjunction with air-swept grinding mills, more particularly roller mills, in which case the separator is an integral feature accommodated in an upward extension of the mill housing itself. Alternatively, the material-laden air from the mill can be fed through a riser duct to the separator, which can thus be used as an independent device for the separation of coarse particles from a stream of air carrying dust or other particulate matter. Its mode of operation, and the subsequent precipitation of the fine particles from the carrying air, are as already described.

2.2.3 Circulating air separators

The circulating air separator (as a generic designation) differs from the static separator and the bladed rotor separator in some important respects:
- the material for classification is fed mechanically to the separator by means of a suitable continuous conveyor;

- the air current required for the functioning of the separator is generated by a fan inside the separator casing or mounted outside it;
- the material for classification is introduced into the stream of air by means of a distributing disc or similar device.

Circulating air separators are the most extensively used type of classifying equipment for fine particles in the cement industry. They comprise machines differing widely in their design features, but nevertheless embodying the same basic operating principles, the differences being confined to the method of material feed and distribution and of controlling the performance of the separator. It would be outside the present scope to attempt a description of the many variants offered by manufacturers. Only two main types will be considered: the conventional mechanical air separator (or centrifugal separator) and the now increasingly used cyclone air separator.

Conventional air separator (Fig. 30c)

An air separator of this general type comprises an outer casing, an inner casing (the upper part of which forms the separating or classifying chamber), a ring of guide vanes, the distributing disc or plate, the main circulating fan and the auxiliary fan (the latter known also as counter-vanes or secondary blades in some manufacturers' literature).
The main fan, which functions as a radial fan, produces a circulating air current in the separator. It flows upwards in the inner casing and downwards in the space between this and the outer casing, re-entering the inner casing through the ring of

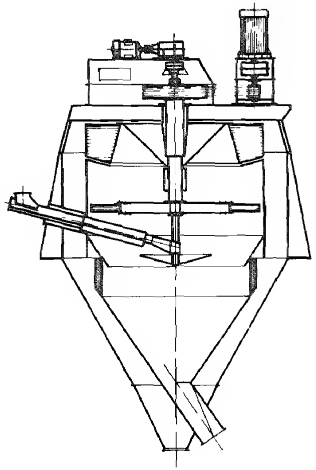


Fig. 30c: Conventional air separator

fixed guide vanes. The upward current of separating air flows past the distributing disc and through the rotating blades of the auxiliary fan.
The pulverized material for classification is fed onto the distributing disc and flung outwards by centrifugal force. Large particles collide with the wall of the inner casing and fall into the tailings outlet in the conical bottom part of this casing. The smaller particles remain airborne and are carried upwards to the auxiliary fan, which accelerates the air which has already acquired a spiralling motion in the ring of guide vanes. The centrifugal forces set up in this way fling the coarser particles against the wall of the inner casing, so that these, too, are discharged through the tailings outlet. The finer particles continue upwards in the air current and are drawn through the blades of the main fan, which further accelerates the air and discharges it into the product collecting chamber, i. e., the space between the inner and the outer casing. Here the fine particles are precipitated from the downward spiralling air, as in an ordinary cyclone, and pass out of the fines outlet in the conical bottom part of the outer casing.

Control possibilities·
The separation characteristic of the air separator can be modified in various ways

(1) Adjustments to the auxiliary fan:
Adjustments to the auxiliary fan will affect the spiralling air current and therefore modify the centrifugal acceleration of the particles carried in the air, so that a shift in cut size is obtained. At the same time, for a given performance characteristic of the main fan, the circulating flow rate and flow velocity will also be altered in consequence. Whatever type of control intervention is applied, the aim should be to obtain the requisite centrifugal acceleration for size separation with the least possible pressure drop.
Some possibilities for the control of various types of air separator.

(a) The speed of the auxiliary fan and the performance of the main fan are constant:
- Without alteration of the auxiliary fan blade angle setting: Increasing the number or size of the auxiliary fan blades shifts the cut size to a smaller particle size. If there is scope for radial adjustment of these blades, a reduction of the clearance between them and the wall of the inner casing will produce a similar shift in the cut size. Conversely, reducing the number or size of the auxiliary fan blades, or increasing their clearance, will result in increased particle cut size.
- Alteration of the auxiliary fan blade angle setting: Maximum acceleration of the spiral air current is obtained with the auxiliary fan blades set vertical. Any adjustment up to an angle of 45° on either side reduces the effective (projected) blade surface area and thus also the radial acceleration. If the angle is further increased with respect to the vertical, a fan effect propelling the flow of air is developed, while the radial acceleration is further reduced. If the auxiliary fan blades are sloped in their direction of rotation, they will strengthen the air flow due to the main fan, and the cut size will be increased. If the blades are sloped in the opposite direction, they will reduce the flow, and the cut size will be decreased.

(b) Adjustment of the auxiliary fan speed while the performance of the main fan remains unchanged:
- The change in speed changes the acceleration imparted by the auxiliary fan to the spiralling air current. If the available speed control range for this fan is insufficient for the required purpose, the measures indicated under (a) may additionally be applied.

(2) Adjustments to the main fan:
The air flow velocity, and therefore its capacity to carry along the particles of material and keep them airborne, is affected by changing the performance of the main fan. Depending on the design of the air separator, the necessary adjustments may be performed while the machine is running or may require it to be stopped. More particularly, the following adjustment possibilities are available:

- changing the speed of the main fan,
- changing the effective surface of the fan blades;
- reducing the intake cross-section of the fan impeller by means of adjustable louvres;
- adjusting the setting angle of the guide vane ring.

In most air separators there is scope for the interlinking of several control interventions, so that the separation characteristic can be modified to suit a wide range of operating conditions.
In particular cases, e. g., for the optimization of plants operating under high load or where special requirements have to be fulfilled by the classified material, it may be necessary to carry out extensive investigations and tests in order to ascertain the most favourable and most economical setting of the separator to meet these conditions.
In general, it is preferable not to regard the air separator as an individual piece of equipment, but to consider it in combination with the grinding plant with which it has to interact.
The quality of the separating effect depends not only on the technical design features, but also on the operating load of the separator, and attains its optimum within the design performance range. Outside this range the quality declines. This being so, wrong conclusions may be drawn if the separator is considered in isolation from other equipment. For instance, if some fault in the mill causes poorer size reduction, the circulating load in the closed grinding circuit and therefore the operating load of the air separator will increase, a situation that could incorrectly be interpreted as being due to a decline in separator performance.

Cyclone air separator (Fig. 30d):
In terms of design features and classification principle the cyclone air separator is basically similar to the conventional type of separator.
The differences consist in the external arrangement of the air circulating fan and product collecting cyclones.
The fan, which is characterized by better efficiency and can develop higher pressures than the slow-running internal fan of the conventional air separator, enables the fines to be precipitated from the air in high-efficiency cyclones.

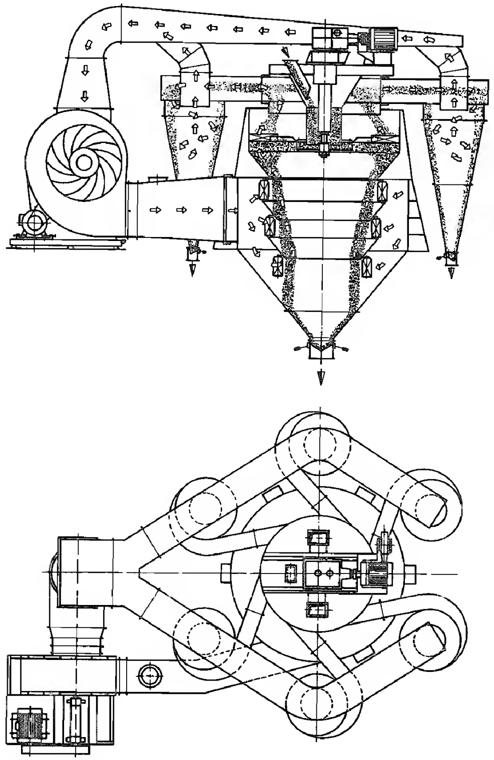


Fig. 30d: Cyclone air separator (O. & K.)

Although this mode of operation involves greater pressure losses, it is more effective than the classifying action achieved in the conventional separator. The separating air circulated through the system is thus very largely relieved of its load of fine particles before being returned to the separating chamber.
In the conventional separator with its less effective separating action there occurs internal recirculation of more particularly the very fine fractions, resulting in diminished classification performance. This snag is virtually eliminated in the cyclone air separator. As a rule, such separators can therefore be operated with lower rates of air circulation than conventional ones of comparable throughput capacity and can achieve better separation in the very fine particle size range.

Cyclone air separators are therefore very suitable in connection with the manufacture of high-strength and very high-strength cements, i. e., ground to a high degree of fineness. Also, the flow rate of the separating air circulating through the cyclone air separator can be varied within wide limits, enabling not only the specific surface but also the particle size distribution of the finished product to be controlled.
On account of the higher air rates and more favourable size classification performance, these separators can be operated at higher specific separating chamber loadings than is practicable with conventional separators. It is particularly the higher air rates and the scope for varying them over a wide range that makes the separator performance much less sensitively dependent on the loading of the separating chamber, i. e., even quite large variations in the loading do not result in any major decline in the quality of performance.
As in conventional air separators, the material for classification may also be dried or cooled in the cyclone air separator.

Control possibilities:
In principle, the possibilities for the control of cyclone air separators are the same as those of conventional ones, except of course that, as already noted, the air flow rate can be varied within a much wider range. For normal purposes, i. e., the production of raw meal and cement in their usual degrees of fineness, the scope for control provided by changing the auxiliary fan speed and/or the air flow rate is usually adequate.

2.2.4 Channel wheel separator

In this machine, which differs radically from the separators described so far, the material for classification is fed from above through a central tube which delivers it to the centre of the channel wheel, a horizontal rotor comprising a series of radial feed channels alternating with extraction channels. Rotation of the wheel hurls the material outwards. Under the action of the Coriolis force it is spread in a thin layer on the rear walls (in relation to the direction of rotation) of the feed channels. At the perimeter the material issues from each channel by streaming out over the throw-off edge of the wheel. Around the circumference, behind each feed channel, is an intake opening through which the air is sucked in and flows radially inwards (Fig. 31).
The actual separating or classifying action is accomplished directly in front of these intake openings. The stream of material coming out of the feed channels is intersected by the air flowing into the extraction channels, so that the trajectories of the particles undergo varying amounts of curvature depending on the size and weight of the particles and resulting from the combination of the inertia forces and the force exerted by the stream of air. The finer particles, whose trajectories are so strongly curved by the air current that they are sucked into the extraction channels, are carried along inwards and into a collecting duct which delivers them to a product collecting cyclone. On the other hand, the trajectories of the coarser particles are less strongly curved. These particles are thus carried out of the intake range of the extraction channels, impinge upon the wall of the separator casing and fall into the tailings outlet (Fig. 31 a).

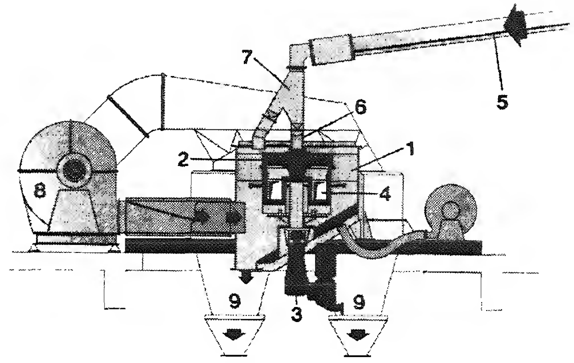


Fig. 31: Channel wheel separator (Krupp-Polysius)
1 separator casing, 2 channel wheel, 3 drive, 4 plenum chamber, 5 feed, 6 feed downpipe, 7 foreign bodies removal, 8 air fan, 9 fines collecting cyclones

Control possibilities:
Experience so far gained with this separator indicates that the cut size can most advantageously be altered by varying the rotation speed of the channel wheel.

Field of application:
As the channel wheel separator is a fairly new development, a comprehensive body of operating experience is not yet available. Results so far obtained would suggest that it can most suitably be used in the manufacture of very high-strength cements and for special size classification purposes.

2.3 Classification in wet grinding

As in dry grinding, so also in wet grinding it is possible to obtain better economy by closed-circuit operation, i. e., by separating the mill product into fines and oversize, the latter being returned to the mill for further grinding. Efforts to make the wet process of cement manufacture as efficient and economical as possible within the currently attainable limits have, among other improvements, led to the development of a procedure using high solids concentrations (up to 1250 g/litre) in the slurry.
Conventional gravity classifiers, such as rake, screw, bowl and up-current classifiers, which function satisfactorily only with considerably lower concentrations than these, are unsuitable for the purpose. It was therefore necessary to devise other methods. Developments in that direction resulted in today's hydrocyclones and curved screens, including the DSM screen.

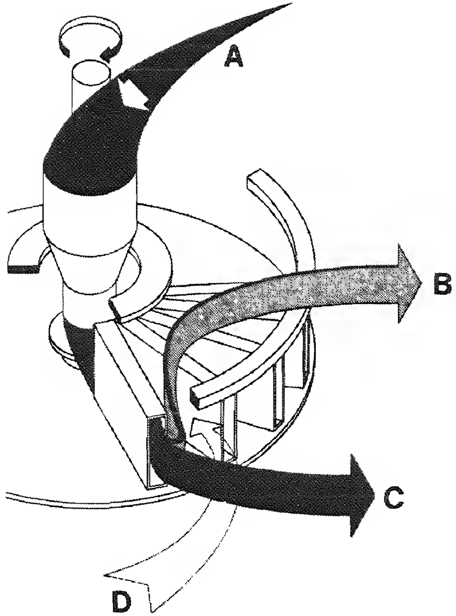


Fig. 31 a: Channel wheel separator: operating principle (Krupp-Polysius)
A feed, B fines, C tailings, D separating air

2.3.1 Hydrocyclones

Along with conventional dry cyclones, hydrocyclones come within the general class of centrifugal separators or precipitators, and the two types have features in common (Fig. 31 b). The raw slurry is admitted under pressure through a tangential nozzle into the upper cylindrical part of the hydrocyclone and is compelled to flow in a downward-spiralling path. The centrifugal forces developed in this way classify the material particles according to mass. The larger and heavier particles are forced outwards and travel in a descending path along the wall of the casing and are discharged from the bottom outlet (apex nozzle). On the other hand, the finer particles are carried upwards in the central part of the swirling flow set up by the throttling action developed in the narrow bottom of the hydrocyclone and are discharged through the central pipe (vortex nozzle) at the top. As the bottom discharge flow contains mainly coarse particles, its solids concentration is substantially higher than that of the top discharge flow. In practice the not very

selective separating action of the hydrocyclone often necessitates the use of two stages of classification, the slurry being passed successively through two hydrocyclones connected in series.
In order to obtain reasonably satisfactory sharpness of separation at a cut size of 0.1 – 0.2 mm when operating with high-viscosity slurries with solids concentrations of about 1000 – 1250 g/litre, large centrifugal forces are required. These are achieved in small-diameter cyclones into which the slurry is fed at relatively high pressures, generally above 2 atm. (gauge pressure). As a rule, hydrocyclones range in diameter from 10 mm (used in multiple assemblies, so-called multicyclones) to about 600 mm.
Against the advantage of technical simplicity of wet grinding systems must be set some drawbacks:

– The particle cut size and sharpness of separation are considerably affected by the slurry feed rate, solids concentration, viscosity and admission pressure;
– Under normal operating conditions these variables cannot always be maintained at favourable values without additional arrangements, e. g., return of part of the product flow to the pump sump or other such measures.
– For the subsequent stages of the raw material preparation process – dewatering or partial dewatering of the slurry and burning it in the kiln – the up to 10% higher water content of the fine slurry discharged from the hydrocyclone

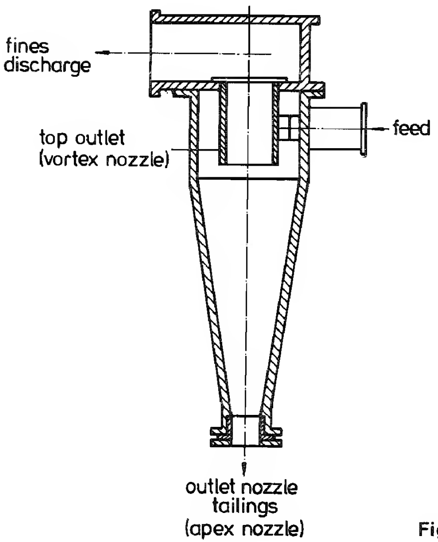


Fig. 31 b: Hydrocyclone

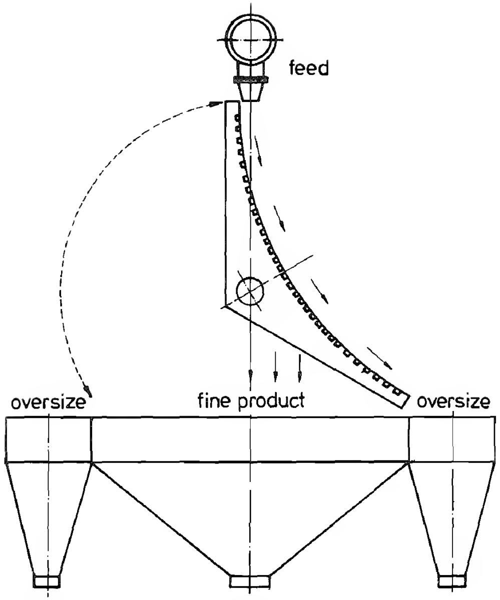
as compared with the raw slurry fed into it entails extra cost in handling and treating this liquid with its lower solids concentration. Also, the hydrocyclones, pumps and pipelines are subject to heavy mechanical wear on account of the high operating pressures and flow velocities.

Control possibilities:
The separating characteristic of a hydrocyclone is governed by its design features, the variable properties of the feed material, the feed pressure and the operating conditions. Influencing factors are:
(1) those determined by the design of the equipment:
– diameter;
– angle of taper of the conical section;
– ratio of the bottom outlet (apex nozzle) and top outlet (vortex nozzle) diameters (in some hydrocyclones this can be varied);
(2) those determined (and modifiable) by the operating conditions:
– feed rate;
– solids concentration in the raw slurry; viscosity (can be modified by the addition of thinning agents);
– feed pressure.

Determining the settings for optimum selectivity and cut size often involves protracted trial-and-error procedures because the interactions of these various factors are very difficult to gauge, even though they are linked by physical relationships.
The separating performance of hydrocyclones responds very sensitively to even quite minor changes in the above-mentioned factors, and in order to obtain a suitably uniform finished product it is essential that the items of equipment connected before and after the cyclones are functioning properly.

2.3.2 Curved screens

In the classification devices so far described, the separating effect is achieved by the action of gravity and/or centrifugal force upon the particles, i. e., the mass or weight of the individual particle determines the magnitude of the force thus exerted on it. On the other hand, in the devices described in the present section, separation is based on the repeated comparison of the size of the particles with a particular aperture (or the geometric projection thereof). Accordingly, the size classifiers in this general category are designated as "screens" (Fig. 31 c).
In principle, a curved screen consists of a grid consisting of horizontal bars of wedge-shaped or trapezoidal cross-section which are arranged so as to form a curved surface. The raw slurry for classification is fed under pressure from above, in a thin stream, onto the screening surface, on which it flows downwards. At the edge of each bar that it encounters a thin layer is "peeled off" from this flow. At the same time, a separation in particle size is effected at these screen bar edges. In much simplified terms it can be said that particles whose centres are above the edge of a bar which they encounter are carried along by the flow of slurry in its further descent along the curved surface, while particles whose centres are below the edge are discharged through the screen.



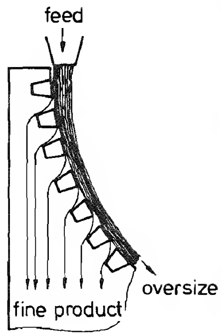


Fig. 31 c: Curved screen (schematic)

On the curved screens used in the cement industry, which have a curvature radius of about 0.5 m and operate with flow velocities of 3–8 m/second, layers corresponding in thickness to about 25% of the aperture width are skimmed off at the successive bars. With the usual apertures employed on these screens, from 0.3 to 1 mm, cut sizes of 0.15–0.5 mm are obtained.
The edges facing the oncoming flow are subject to heavy wear and become blunted in course of time, as a result of which the cut size is shifted to smaller particle sizes. On the other hand, the rear edges of the bars become gradually sharpened by the abrasive action of the particles. By reversing the bars from time to time it is possible, despite increasing wear, to maintain the desired cut size of the screen.
Like hydrocyclones, curved screens can be successfully operated with slurries containing solids in concentrations up to 1250 g/litre, and they have the advantage over hydrocyclones in that the increase in water content of the fine slurry is less pronounced. In many instances, combinations of hydrocyclones with curved screens as secondary classifiers are employed.

Control possibilities:
For a given aperture width between the screen bars the cut size and the selectivity of the screening operation can be modified by varying the flow velocity and solids concentration of the slurry. Reduction of the velocity increases the cut size and the probability that oversize particles will be present in the fines, while the probability of undersize particles being present in the coarse rejects is correspondingly reduced. Increasing the flow velocity produces the opposite effects.
For equal apertures and flow velocities an increase in the solids concentration of the slurry fed to the screen reduces the cut size, but the sharpness of classification becomes poorer. Conversely, with lower solids concentration in the feed slurry there is an increase in cut size and an improvement in sharpness.
As in the case of hydrocyclones, it requires some trial and error to determine the optimum settings for flow velocity and solids concentration for a given purpose. Curved screens likewise respond very sensitively to changes in these parameters, while it is equally essential that the equipment installed before and after the screens should function reliably.

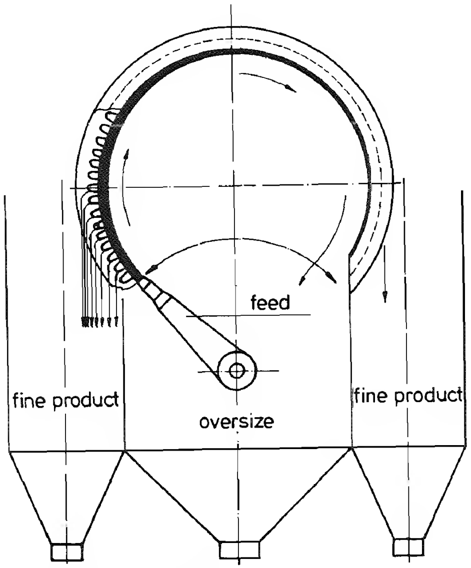


Fig. 31 d: DSM screen (schematic)

DSM screen

The DSM screen (developed by the Dutch State Mines) is a further development of the curved screen and functions on the same principle (Fig. 31 d). In this case the screen surface also consists of horizontal bars of trapezoidal or wedge-shaped section, but here arranged as a cylinder segment comprising an arc of 270°. The raw slurry to be classified is fed tangentially under pressure through a nozzle onto the inner surface of the screen. It rises in a curved path and travels round this surface. On the way, the slurry containing the fine particles is discharged through the apertures between the bars and emerges on the outside of the screen and is collected there. Meanwhile the coarse particles discharged from the inner surface after travelling completely round it are collected in another hopper. The one-sided wear of the bars, which affects the cut size of the classifying operation, is compensated by reversing the direction of flow round the screen. The nozzle can be swung over to the other edge for this purpose.
As compared with ordinary curved screens, the DSM screen attains better sharpness of separation because of the longer path that the slurry travels along the curved surface.

2.4 Criteria for the assessment of classification processes

In the cement industry, classifiers are used mainly in conjunction with closed-circuit grinding. Their operation considerably affects the grinding process. Various criteria and characteristics are used for assessing their performance.

Notation:

A	= classifier feed rate [t/hour] = total mill throughput
F	= fines output rate [t/hour] = finished product output
G	= tailings output rate [t/hour] = circulating load
a,f,g	= undersize (proportion below a certain size) in A,F,G [% by weight]
$\Delta a, \Delta f, \Delta g$	= proportion of a particular size fraction of A, F, G [% by weight]
v_F	= percentage output of fines [% by weight]
v_G	= percentage output of tailings [% by weight]
u	= recycle ratio
p	= precipitation efficiency [%]
t	= selectivity [%]
d_t	= cut size [micron].

Fundamental equations:

$$A = F + G$$
$$A \cdot a = F \cdot f + G \cdot g$$
$$A \cdot \Delta a = F \cdot \Delta f + G \cdot \Delta g.$$

Fines output

The fines output v_F is the percentage (by weight) of the feed material that is separated as fine product in the classification process.

$$v_F = \frac{F}{A} \times 100\ [\%]; \quad v_G = \frac{G}{A} \times 100\ [\%]; \quad v_F + v_G = 100\ [\%].$$

As a rule, in actual plant operation it is impracticable to determine the flow rates of the feed (A), fines (F) and tailings (G) directly by weighing. Therefore the output is calculated from the results of the particle size analysis:

$$v_F = \frac{a-g}{f-g} \times 100\ [\%] \quad \text{and} \quad v_G = \frac{f-a}{f-g} \times 100\ [\%].$$

Errors of measurement inevitably affect the result of the outputs calculated by means of these formulas. A simplified empirical formula, which is sufficiently accurate for practical purposes, has been given by Koulen:

$$v_F = \frac{S_a - S_g}{S_f - S_g} \times 100\ [\%],$$

where S_a, S_g and S_f denote the totals of the undersize percentages obtained in the particle size analysis of the samples of feed material, fines and tailings taken in a test on a classifier. Obviously, only values which correspond to the same particle sizes should be used for determining the totals (see Table 1).

Recycle ratio

The recycle ratio u is the ratio of the classifier feed rate to the fines discharge rate:

$$u = \frac{A}{F} = \frac{100}{v_F} = \frac{S_f - S_g}{S_a - S_g}.$$

This ratio provides a criterion for the loading of the classifier and thus also of the grinding plant. In a grinding plant operating under steady conditions (equilibrium) the rate of feed of new material to the plant must be equal to the rate of fines discharge from the classifier, i. e., the material removed as finished product from the grinding circuit. The recycle ratio can therefore be calculated from the measured values of the rate of new material (M) fed to the grinding plant and of the rate of tailings discharge from the classifier:

$$u = \frac{M + G}{M} = \frac{A}{F}.$$

Table 1: Determination of fines output by Koulen's method (10); tests on a cyclone air separator with 5.2 m diameter (O. & K.)

method of analysis	particle size [µm]	a [%]	f [%]	g [%]
sedimentation	1	3,7	7,5	2,4
	2	6,6	13,0	3,9
	4	11,8	23,7	6,2
	8	17,4	38,7	8,4
	16	27,2	62,6	10,8
	32	48,5	91,8	22,8
	64	79,1	99,3	75,2
air-jet sieving	90	90,0	99,8	85,6
	200	98,7	100	98,1
	–	383,0	536,4	313,4

$$v_F = \frac{S_a - S_g}{S_f - S_g} \cdot 100 = \frac{383{,}0 - 313{,}4}{536{,}4 - 313{,}4} \cdot 100 = \underline{31{,}2\ \%}$$

$$v_G = 100 - v_F = \underline{68{,}8\ \%}$$

$$u = \frac{100}{v_F} = \underline{3{,}21}$$

Precipitation efficiency

The precipitation efficiency of the classifier is:

$$p = \frac{F \times f}{A \times a} \times 100 = v_F \times \frac{f}{a}\ [\%].$$

The precipitation efficiency is referred to a certain given particle size (and varies with the size considered). It denotes what percentage of the material finer than the reference particle size in the classifier feed is discharged as fines.

Selectivity

The selectivity (separation efficiency) is characterized by what percentage of a given particle size in the classifier feed is discharged in the tailings. Plotted against particle size, it appears as a curve (separation curve, Tromp curve) representing the so-called classifier selectivity function (Fig. 32a).
It is not possible to determine the selectivity for a given particle size directly from the size analysis values, as these relate to size fractions, not individual sizes. First, a

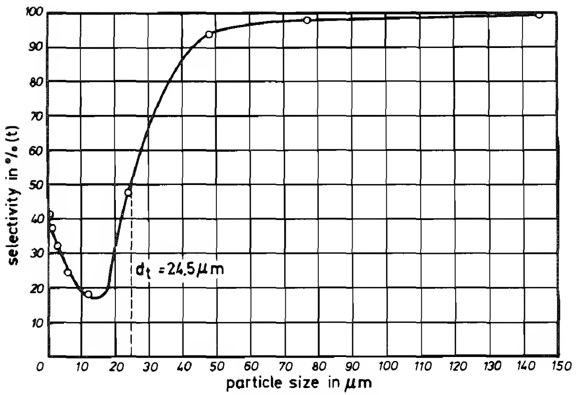


Fig. 32a: Separation curve for cement with a specific surface of 3980 cm²/g (plotted from Table 2)

stepped diagram corresponding to these functions must be drawn and then the average smooth separation curve approximating to that diagram.

The selectivity can be calculated from.

$$t = \frac{G \cdot \Delta g}{A \cdot \Delta a} \times 100 = v_G \times \frac{\Delta g}{\Delta a}\ [\%].$$

Cut size (Table 2)

The cut size is defined as that particle size d_t of which half the particles are discharged in the fines and half in the tailings, i.e., for that size the selectivity is 50%.
If sharpness of separation is poor, more than 50% of even the finest particle sizes may end up in the tailings, so that then no definite cut size d_t exists.

Specific precipitation efficiency

It is possible to assess the performance of a classifier without having to determine the separation curve. For that purpose the particle size distribution and the specific surface areas of various particle size ranges in the classifier feed material can be considered instead.
The sieve undersize amounts (percentages by weight passing the respective test sieves) are plotted against the particle sizes, as has been done in the upper diagram of Fig. 32b. In the lower diagram the Blaine values corresponding to these

Table 2: Calculation of selectivity values; tests on a cyclone air separator with 5.2 m diameter (O. & K.)

particle size	fines from classifier $V_F = 31{,}2\,\%$			tailings from classifier $V_G = 68{,}8\,\%$			feed to classifier	selectivity
	f	Δf	Δf · V_F/100	g	Δg	Δg · V_G/100	$\Delta a_{err.}$	t
[µm]	[%]	[%]	[%]	[%]	[%]	[%]	[%]	[%]
			(3) · 31,2/100			(6) · 68,8/100	(4) + (7)	(7) · 100/(8)
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
1	7,5	7,5	2,34	2,4	2,4	1,65	3,99	41,3
2	13,0	5,5	1,72	3,9	1,5	1,03	2,75	37,5
4	23,7	10,7	3,34	6,2	2,3	1,58	4,92	32,1
8	38,7	15,0	4,68	8,4	2,2	1,51	6,19	24,4
16	62,6	23,9	7,46	10,8	2,4	1,65	9,11	18,1
32	91,8	29,2	9,11	22,8	12,0	8,26	17,37	47,6
64	99,3	7,5	2,34	75,2	52,4	36,05	38,39	93,9
90	99,8	0,5	0,16	85,6	10,4	7,15	7,31	97,9
200	100	0,2	0,06	98,1	12,5	8,60	8,66	99,3
400	100	–	–	100	1,9	1,31	1,31	100
	–	100	31,21	–	100	68,79	100	–

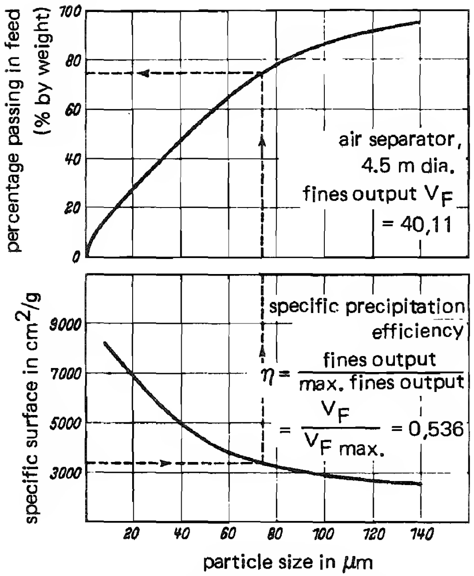


Fig. 32b: Determining the maximum percentage output of fines v_{Fmax} for a given classifier feed material (15)

undersize amounts have been plotted. For this purpose the Blaine values have to be determined for a sufficient number of different size fractions to enable a continuous curve to be drawn.

Suppose that the desired classifier product must have a Blaine value (specific surface) of 3300 cm²/g, as in the example represented in Fig. 32b. In the lower diagram this is found to correspond to a particle size of 75 microns in the classifier feed. Projecting this value perpendicularly upwards into the upper diagram shows this size to correspond to about 75% passing the sieve (= v_F max.). This fines output v_F max. = 75% represents the highest attainable output of finished product having a specific surface of 3300 cm²/g if the classifier feed is separated ("cut") with complete sharpness at 75 microns. Actually, complete sharpness of cut is never achieved. The ratio of the actual fines output v_F (= 40.11% in the example considered) to the highest attainable output v_F max. provides a criterion for the separating performance of the classifier.

Classifier tests

The rate of feed and the particle size distribution of the material supplied to the classifier affect the classification result. Therefore these two parameters should be kept constant during the period of the trials, and the grinding plant should be operating under steady conditions (equilibrium). In order to compensate for any variations in the feed, the samples of the material flow rates A, F and G are taken over periods of 5–10 minutes at close intervals of 1–2 minutes. Gross samples of the three flows – classifier feed, fines, tailings – are respectively prepared and the specific surface values and particle size distributions are determined.
The test record should include the relevant technical data of the classifier, e.g., the speed and setting of the auxiliary fan, the settings of the valves or dampers in the air circulating system, etc., together with particulars of the mill, the mill feed material, and the cooling air or hot gas introduced into the grinding plant in so far as these affect the classification process.

Evaluation of the classifier tests

The undersize percentages (a, f, g) for the classifier feed rate (A), the fines output rate (F) and the tailings output rate (G) obtained in classifier tests are indicated in Tables 1 and 2. The values of a, f and g can advantageously be plotted against particle size in a diagram with linear scales. The values for the percentage output of fines v_F and for the recycle ratio u are calculated by the methods given in Section 2.4. The selectivity values, which are needed for drawing the separation curve, are calculated as shown in Table 2.

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3 Grinding

3.1 General Introduction

By grinding is understood the comminution of materials to a powder. In cement manufacture by the dry process it constitutes the final stage in the production of raw meal (raw grinding). The clinker discharged from the kiln has to be ground to a fine powder which, with the admixture of some gypsum, is the finished product of the whole process: cement. This final clinker grinding operation is often referred to as finish grinding. The terms "pulverizing" and "milling" are basically synonymous with "grinding", but are mostly confined to the comminution of coal or lignite for use as pulverized fuel.
The object of grinding is, more particularly, to increase the specific surface of the material – while conforming to a desired particle size distribution – to such an extent as to obtain adequate reactivity for the next stage in the cement manufacturing process or adequate reactivity in the finished product (the cement) itself.
In the cement industry about 75% of the total electric energy consumption is consumed in grinding the raw materials, the clinker and, where applicable, the fuel

Because of the diminishing available sources of energy and the attendant rise in energy costs it is important to pay particular attention to this major cost item.
Grinding of materials in the two main types of mill now commonly employed – tube mills and roller mills – inevitably involves very considerable energy losses. The actual energy input required for reducing a given material to a certain particle size far exceeds the energy theoretically needed for breaking down the particles and thus increasing the surface area of the material. Depending on the criteria applied, it is estimated that only between 2 and 20% of the energy supplied to the grinding system is utilized for producing new surface. The remainder, i. e., between 98 and 80%, is lost energy, largely going to waste as heat and vibration.
There has of course been no lack of effort to improve present-day grinding systems and to achieve greater economy in terms of energy utilization. Some positive results have indeed been achieved. Even so, the attainable improvements in this respect are still only a mere fraction of the energy losses associated with grinding.
Further developments – e. g., based on fundamentally different comminuting actions such as those of the centrifugal forces in planetary ball mills or of the pressure waves associated with electrical discharges – do indeed open up some interesting prospects, but such processes are still very much in the experimental stage and nowhere near full development for use on an industrially meaningful scale.
The problem facing the operator of a grinding plant is to decide how to achieve maximum economy with the equipment now available. For this it is essential to know the present-day possibilities and limitations and to be properly informed of the ways and means of assessing the performance of the plant. Besides, the improvement of existing systems and the design of new ones have to be based on a sound understanding of these principles.
The energy required for the comminution, or size reduction, of a material to a certain required fineness (characterized by the specific surface of the product obtained) will depend on the hardness of the material, its compressive strength, its brittleness (or its elasticity or its plasticity), the size and shape of its particles, its temperature and moisture content, and of course also on the nature of the comminuting action exerted by the grinding process employed These factors in combination determine the resistance that the material offers to size reduction and can be regarded as specific of the material.
This specific resistance to grinding can be expressed as specific energy requirement and can provide a criterion for directly comparing the size reduction properties of different materials with one another.
The size reduction of a material – in the cement industry the materials concerned are minerals or mixtures of minerals – therefore involves overcoming specific resistances or forces. In the main these are crystal bonding forces and interfacial bonding forces. In crystalline materials, fracture is initiated at flaws which are always present in such materials and which constitute weak spots that impair the homogeneity of the crystals. When the material is subjected to load, these flaws act as notches where stress concentrations occur and where fracture will be initiated when the stresses exceed the local strength of the material.

The probability of fracturing is governed not only by the magnitude of the loading, but also by the rate (speed) of load increase because it is this that determines whether the material will, within limits, behave in a more plastic or in a more elastic manner.
The efficiency of a size reduction process may be judged by comparing the energy consumption of an industrial grinding plant with the energy theoretically required for achieving the size reduction on the basis of the physical theories of particle fracture. The actual energy consumption is always found to be many times greater than the theoretical value, the difference between the two values being an indication of the energy lost or wasted in grinding the material.

More particularly, these energy losses are due to:
(1) Friction between the particles of the material themselves and between them and the grinding elements (grinding media, liner plates, grinding rollers, grinding bowl, etc.). In ball mills there is moreover friction between the grinding media themselves and between them and the mill lining. The friction is converted into heat, noise and electrostatic charge.
(2) Wear of the grinding elements; elastic and, to some extent, plastic deformation of the elements.
(3) Elastic deformation of the material to be comminuted until the fracturing stress is attained at the flaws and weak spots in its particles, so that these break up.
(4) Plastic deformation of the material to be ground.
(5) Formation of particle agglomerations.

3.2 Forms of comminuting action

The conventional machines for the size reduction, or comminution, of materials make use of the following types of mechanical action applied to the particles: compression, shear, percussion and impact. As a rule, there are no clear-cut divisions between these various actions, and in most machines two or more of them occur simultaneously, i. e., the particles are subjected to a combination of actions.

3.3 Types of grinding mill

3.3.1 Tumbling mills

In machines of this category the size reduction of the feed material is accomplished by the action of gravity upon the contents of the mill – which is usually a tube-shaped or drum-shaped unit rotating on a horizontal axis – in the course of its rotation. In the cement industry such mills are used for raw material, coal and clinker grinding.
A distinction is to be made between tumbling mills containing grinding media, usually consisting of steel balls, and those containing no grinding media (or only a small quantity), the comminuting action being performed mainly by the feed material itself, i. e., the coarse particles act as their own grinding media in tumbling upon, and rubbing against, one another (autogenous mills).

Tumbling mills operated with grinding media are usually of the type called tube mills.

3.3.2 Tumbling mills with grinding media (tube mills)

In this very extensively applied grinding system the grinding media and the feed material to be ground are brought together in a rotating tubular or drum-shaped compartment. The media and material are lifted some distance at the rising side of the mill in its rotational motion and, after reaching a certain height, come tumbling down (cascading and/or cataracting). The actual height to which they are lifted depends on a number of factors· the speed of the mill, the type of lining, the composition and shape of the grinding media, the filling ratio (mill loading percentage), and the properties of the mill feed material.
Size reduction work is done both during the rising movement and during the subsequent cascading/cataracting of the mixture of grinding media and feed material. In the first part of this cycle, i. e., the lifting stage, the material is reduced mainly by compressive and shearing action. Then, in tumbling back to the bottom of the mill, it is subjected mainly to impact and percussion.
The grinding media used in the cement manufacturing industry are nearly always steel balls or short cylindrical steel media (Cylpebs). Porcelain or rubber-jacketed steel balls or porcelain Cylpebs are used only for exceptional purposes, e. g., in the production of white cement. Grinding media consisting of other materials, such as flint, may be used for the reduction of very soft raw materials, e. g., chalk.
The mill is lined with plates, usually of steel and commonly referred to as liners, which serve to protect the mill shell against wear and also to assist the lifting of the feed material/grinding media mixture. In wet grinding, linings made of rubber or a combination of rubber and wood may be used. These materials, and also porcelain linings, are employed in white cement manufacture.
A third function – besides providing wear protection and helping to lift the mill contents – that the liners are sometimes required to perform is that of "classifying" the grinding media according to size along the length of the mill. This effect is achieved by the use of specially shaped liners.

3.3.3 Various forms of construction for tube mills

The design features of certain types of tube mill which are of little or no importance in connection with cement manufacture (e. g., rod mills, trommel screen mills, etc.) will not be described here.
The technical nomenclature applied to tumbling mills tends to be inconsistent. For instance, the designation "ball mills" is sometimes rather loosely applied as a generic term to describe all these mills (except rod mills, autogenous mills, etc.) or, alternatively, this term is confined to such mills with a low length/diameter ratio (below 3·1 or 2:1). The latter is a rather arbitrary distinction, while the designation "ball mills" may be misleading because the grinding media are not necessarily spherical, but may be cylindrical bodies such as Cylpebs. Mills characterized by a length/diameter ratio of 3:1 or more are conventionally called "tube mills". It

would, however, be more logical to apply this designation to this whole class of mills, irrespective of the length/diameter ratio.

Tube mills can be classified according to various criteria.
(1) number of grinding compartments:
– single-compartment mills;
– multi-compartment mills;
(2) method of product discharge:
– end discharge through mill bearing trunnion;
– end discharge through trunnion with stream of air (air-swept mills),
– end discharge at periphery of mill;
– central discharge at periphery of mill;
(3) nature of the grinding process:
– wet grinding:
  in open circuit;
  in closed circuit;
– dry grinding:
  in open circuit;
  in closed circuit (with air classifier equipment).

3.4 Motion of grinding media in tube mills

Rotation of the mill causes the charge consisting of grinding media and feed material to be lifted some distance by friction between the media and the lining. The height to which the charge is lifted will depend on a number of factors.
– circumferential velocity of the mill;
– shape, size and weight of the grinding media,
– friction between the lining and the grinding media; its magnitude can be modified by design features of the liners;
– friction within the mill charge itself; the magnitude of these frictional forces is in turn governed by the loading percentage, the proportion of feed material in relation to grinding media, and the properties of the material, such as its moisture content and flowability.

It is not possible to quantify all these variables and establish an exact mathematical analysis. To simplify the problem of grinding media motion, the behaviour of just one of them – say, a ball – will first be considered.
The ball is subjected to centrifugal force (due to the rotation of the mill) and to the force of gravity. Under the combined action of these forces the ball will travel in a circular path, i. e., in contact with the rising wall of the mill, so long as the radial component $m \times g \times \cos \alpha$ of the gravitational force is less than the centrifugal force $\frac{m \times v^2}{r}$. At the point of the circumference where the radial component of the gravitational force becomes larger than the centrifugal force, the ball detaches itself from the wall and falls back into the mill. In doing this it travels along a parabolic path (Fig. 33).

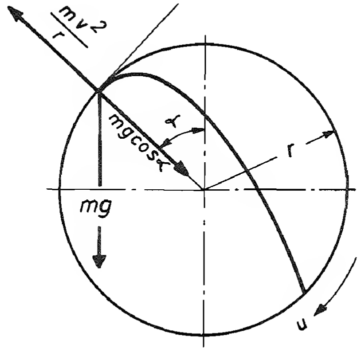


Fig. 33: Forces acting on a grinding ball

The following notation will be used:

m	mass of the grinding ball	[kg]
v	(circumferential) velocity	[m/second]
r	radius of the circular path	[m]
α	angle of detachment	[degrees]
n	speed of mill rotation	[revolutions/minute]
g	acceleration of gravity	[m/second^2].

On the assumption that the ball cannot slide or roll on the mill lining and that it therefore moves at the same angular velocity as that of the mill shell, the angle of detachment can be determined from the equilibrium condition

$$m \times g \times \cos\ \alpha = \frac{m \times v^2}{r},\ \text{giving:}\ \cos\ \alpha = \frac{v^2}{g \times r},\ \text{while}\ v = \frac{2 \times \pi \times r \times n}{60},$$

so that $\cos\alpha = 1.118 \times 10^3 \times r \times n^2$. Above a certain rotational speed – the so-called critical speed – detachment of the grinding media will not occur, i.e., they will be carrier round and round the circumference. Hence $\cos\alpha = 1$. This speed is characterized by the condition that the centrifugal force and the gravitational force at the top of the circumference are in equilibrium, so that: $\frac{m \times v^2}{r} = mg$.

Putting $v = \frac{2 \times \pi \times r \times n}{60}$, we obtain for the critical speed.

$$n_{crit} = \frac{30}{\sqrt{r}}\ \text{or}\ \frac{42.3}{\sqrt{D_i}}\ [\text{r.p.m.}],$$

where D_i denotes the internal diameter of the mill (in m).

The assumption on which the above calculation is based, namely, equal angular velocity of the grinding media and the mill shell, is not fulfilled under actual operating conditions. Only at rotational speeds substantially higher than the theoretical critical speed will the grinding media remain in contact with the lining all round the circumference.

Hence the critical speed serves merely as a reference value for describing mill speeds, which are often expressed as a percentage of the critical speed.

As has been determined experimentally, the most favourable grinding effect is obtained in the range between 68 and 75% of the critical speed. Tube mills are normally operated within these limits. As a rule, the various manufacturers have adopted certain speed ranges as most suitable for their mills.

The bulk volume of the grinding media charge in tube mills is usually between 20 and 35% of the internal volume of the grinding compartment. This filling ratio is known as the loading percentage or grinding media load of the mill. The media form a bed comprising a number of layers. When the mill rotates, the inner layers detach themselves before the outer ones. If the speed of rotation of the mill is sufficiently high and the loading percentage is appropriately chosen, the media perform a cataracting motion (Fig. 34).

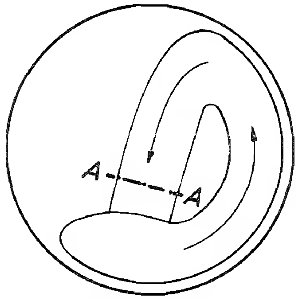


Fig. 34: Cataracting of grinding media

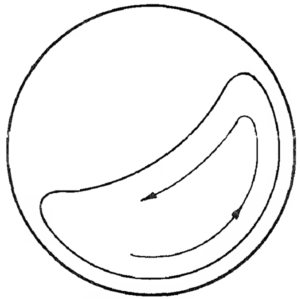


Fig. 35: Cascading of grinding media

The feed material, which is lifted along with the grinding media and is subjected to compression and shear during this part of the motion, is pulverized mainly by impact and percussion in the zone "A", where almost the entire energy of the falling grinding media is concentrated. This form of communuting action is especially effective in the primary size reduction of relatively coarse feed material supplied to the mill.

Under similar conditions, but with higher loading percentages, the grinding media will perform a cascading motion (Fig. 35). In this case the inner layers of the grinding media charge detach themselves before the outer ones, and the latter fall back onto the media which are already detached and moving downwards. As

contrasted with what occurs in cataracting, in cascading the motion of the grinding media in their downward stream is characterized by flowing and rolling rather than falling. Thus the energy of falling is distributed over a larger area and therefore less concentrated. For this reason, cascading is not very suitable for the comminution of coarse feed material, but is on the other hand very effective for fine (secondary) grinding.

For equal circumferential velocity of the mill shell the actual pattern of grinding media motion between the two extremes of cascading and cataracting is governed by a number of factors:
- shape and surface configuration of the liners;
- composition of the grinding media charge;
- loading percentage of the mill;
- resistance of the material to comminution;
- moisture content of the feed material.

Through the first three factors it is possible to modify the motion of the grinding media so as to adapt it to the operating conditions in any given case.

3.4.1 Motion of the material being ground

In tube mills the raw material is fed, and the product discharged, in a continuous flow. In the course of size reduction, the material moves from the inlet to the outlet of the mill. This motion is due to several causal factors. In dry grinding these are:

- increase in bulk volume of the material according as it is ground to finer particle size;
- increasing flowability of the material as it is more finely reduced;
- displacement of the fine material with better flow properties by the coarser feed material with poorer flowability.

In addition, in air-swept mills the stream of air passing through the mill assists the longitudinal progress of the material.

3.4.2 Effect of volume increase on grinding (Fig. 36)

Besides proper adjustment of the grinding media to suit the feed material, the proportion of material in relation to grinding media in the mill is a major factor governing the effectiveness of the grinding process. If the proportion of material is too low, a high percentage of direct impacts between grinding media will occur, so that no material is pulverized between them and no comminuting work is done. On the other hand, if there is too much material in the mill, too much of the energy of falling will be dissipated in displacing the particles from between the impacting grinding media and will thus be wasted.

Experience shows that the best grinding results are generally obtained when, with the mill at rest, the top level of the material coincides with the top level of the grinding media charge along the whole effective length of the mill. In order to obtain such conditions it is necessary that, with increasing bulk volume of the material with progressively finer size reduction along the mill, the speed at which

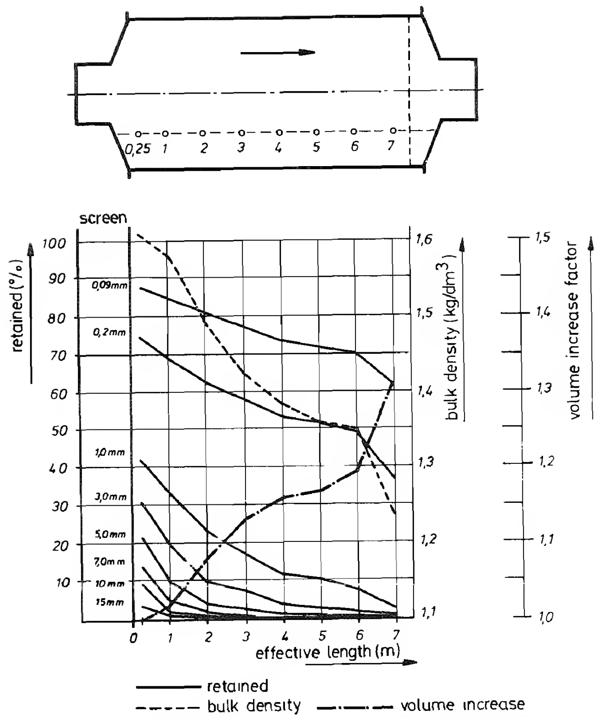


Fig. 36: Increase in volume (bulking) of the feed material with increasing fineness of grinding in a tube mill of 4.0m diameter and 7.5m effective length

the material is transported longitudinally through the mill proportionally increases. This is achieved thanks to the better flowability that the material acquires as it becomes more finely ground.

The level that the material settles down to under continuous mill operating conditions is governed by the composition of the grinding media charge. As a rule, this level can be lowered and the residence time of the material in the mill be shortened by using coarser grinding media (larger balls, etc.), and vice versa. In actual practice the problem is to determine the operating point at which, with maximum throughput, the required degree of size reduction is achieved.

Wet grinding

In wet grinding carried out in tube mills the axial progress of the material is governed mainly by the flow velocity of the slurry, its water/solids content and the fineness of the raw slurry particles. If the water content is appropriately adjusted, classification of the material according to particle size occurs in the mill, which is advantageous because particles already sufficiently reduced in size will then not unnecessarily be further subjected to grinding action.

3.5 Calculating the mill drive power

The power input required for driving a tube mill can be determined from the relationship:
power = torque × angular velocity (Fig. 37).

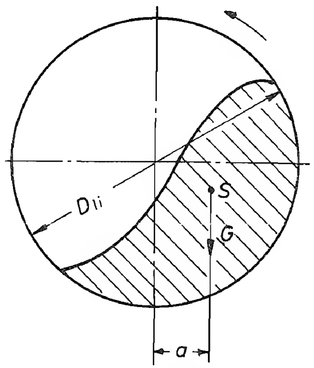


Fig. 37: Simplified geometric relationships for determining the mill drive power

When the mill is in operation, the mass of grinding media and feed material forms an irregularly shaped asymmetrical body whose centre of gravity is at a distance a from the vertical centre-line of the mill cross-section. The opposing torque is thus equal to the weight G of the grinding media multiplied by the distance a, i.e. $M = g \times a$.
Relative to the rotating mill shell, the centre of gravity S, which is at rest, has the angular velocity $\frac{2 \times \pi \times n}{60}$.
From the relation:
power = torque × angular velocity we obtain.

$$N = \frac{G \times a}{102} \cdot \frac{2 \times \pi \times n}{60},$$

in which a is an unknown quantity. By way of simplification it can be assumed that, in all mills with comparable loading percentages and rotational speeds, there is a constant ratio between a and the internal diameter of the mill, so that we can write: $a = X \times D_{Li}$.
On substitution of this relation into the above expression for N we obtain.

$$N = \frac{G \times X \times D_{Li}}{102} \times \frac{2 \times \pi \times n}{60} \text{ [kW]}.$$

To obtain a simpler expression, we can introduce a power factor.

$$C = \frac{X \times 2 \times \pi}{60 \times 102} \text{ so that: } N = G \times D_{Li} \times n \times C \text{ [kW]},$$

where:

N = power consumption [kW]
G = weight of mill charge [t]
D_{Li} = internal diameter of mill [m]
C = power factor
π = speed of mill rotation [revolutions/minute]

This calculation is, as already stated, based on the assumption that in all mills with comparable loading percentages the distance a is a constant proportion of the diameter. Thus no account is taken of any features that affect the lifting height of the mill charge and the magnitude of the distance a, such as the shape of the liners, the type of grinding media, the weight of the grinding media charge, and the physical properties of the feed material.
Values for the factor C have to be determined empirically. Unfortunately, they exhibit a wide range of scatter between the upper and lower limiting sizes of the grinding media employed, so that the power consumption values calculated with the aid of such factors tend to be inaccurate. The power consumption of a tube mill determined in this way, which takes account of the mechanical losses of the mill and drive but not the efficiency of the drive motor, is therefore to be regarded only as an approximate guide value. To allow for the efficiency of the motor an extra 4% should be added (Fig. 37a).

Example

Data of the tube mill:
internal diameter $D_{Li} = 3.13$ m; effective length 11.5 m; grinding media charge 82.5 t; loading 20.5%;
C = 0.252 (coarse grinding media);

rotational speed n = 17.5 r.p.m.;
feed material. raw material for cement manufacture.
Power consumption: $N = 82.5 \times 3.13 \times 17.5 \times 0.252 = 1139$ kW
adding 4% (= 46 kW) gives N = 1185 kW.
This calculated value compares with N = 1195 actually measured for this mill operating under these conditions.

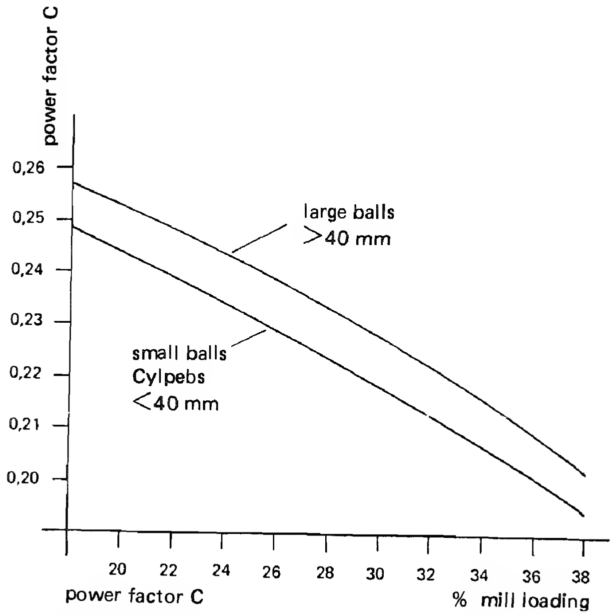


Fig. 37a: Power factor C for determining the drive power of a tube mill

3.6 Tumbling mills without grinding media (autogenous mills)

The motion of the mill charge in an autogenous mill is subject to the same principles as those operating in an ordinary tube mill, except that instead of balls or other grinding media the larger pieces of feed material themselves perform the comminuting function. Both types of mill are governed by the same physical relationships, namely, that the new surface produced is proportional to the energy input to the mill and that the work done in comminuting the material is determined by the mass and height of fall of the grinding media.

Differences in design between tube mills and autogenous mills are due to the fact that in the latter type of grinding machine it is necessary to operate with a larger mill charge volume and greater height of fall of the "grinding media" in order to develop sufficient comminuting energy, because the lumps of material that have to perform this function are of lower density than the steel balls or Cylpebs used in tube mills. The comminution of one piece of feed material by another is more effective if there is a pronounced difference in mass between the two pieces; therefore the particle size distribution of the feed material in an autogenous mill must not be homogeneous along the length of the mill. This requirement is fulfilled by using mills which are short in relation to their diameter and by installing so-called deflectors, which are internal fittings that deflect the material towards the centre of the mill.

Besides requiring larger effective volumes and different length/diameter ratios, autogenous mills must also rotate at higher speeds than tube mills – so as to lift the charge higher and thus obtain greater heights of fall – in order to attain comparable throughput rates. Autogenous mills do indeed differ considerably from tube mills in having very low length/diameter ratios, of the order of only 1:5, and their speeds are in the range of about 70 to 100% of critical. They are used both for drying grinding (Aerofall mills) and for wet grinding.

In dry grinding, the sufficiently pulverized material is removed from the mill usually by a stream of air, which enables the granulometric composition of the product of the mill to be controlled within certain limits. Such air-swept mills demand high air throughput rates, which can be turned to advantage by combining the autogenous grinding operation with drying of moist feed material.

In wet grinding, the product is discharged by overflow through the trunnion outlet or through sieve plates. In comparison with ordinary tube mills containing grinding media, the raw slurry fed to an autogenous mill should have an approximately 7 to 10% higher water content.

With autogenous grinding, whether dry or wet, it is possible to grind "naturally" granular bulk materials or materials that have been suitably pre-crushed, subject to the maximum feed particle size not being too large. Sometimes, on the other hand, specially separated larger pieces of rock are added to act as "grinding media" in what could otherwise be too fine-grained a feed material. For the same reason, large steel balls (up to a loading of about 10%) may be introduced into the mill to assist the autogenous grinding action and compensate for variations in the granulometric composition of the feed material.

In the cement industry, autogenous grinding is used only in certain individual cases and then only for the primary grinding of raw materials. With the currently available mills of this kind it is not possible to obtain a finished product of sufficient fineness to serve as raw meal for kiln feed. This is so because the selective size reduction effect associated with autogenous grinding, which may be advantageous in the preparatory processing of other raw materials, is rather undesirable in raw grinding for cement manufacture. More particularly, it means that homogeneous hard components in the feed material are liable to be inadequately broken down.

The main advantage of autogenous grinding, i. e., without (or with only a limited

proportion of) grinding media, lies in the lower rates of wear in comparison with grinding in tube mills. On the other hand, claims that autogenous grinding involves lower specific power consumption, such as are sometimes put forward in the literature, appear not to be substantiated. In making any such comparisons it is necessary to compare the results of the grinding process not merely on the basis of percentages retained on test sieves, but also in terms of specific surface values. Because of the more highly selective character of the comminuting action, which tends to produce more cleavage along the contact faces of the micro-crystals, the autogenous grinding product tends to contain a lower proportion of very fine particles.

3.7 Monitoring of wear

The internal fittings of tube mills, such as the liners, the feed and discharge devices, the intermediate and discharge diaphragms and the grinding media charge, are all subjected to severe mechanical actions. These manifest themselves in wear of the parts concerned, the degree of wear being dependent on the properties of the feed material to be ground and on the quality (wear resistance) of the materials of which these wearing parts are made. Besides, the ageing effect of mechanical repetitive or cyclic loading on these parts is important.
In order to eliminate as far as possible the occurrence of damage during mill operation and to compensate for decline in mill performance due to wear, it is advisable regularly to inspect the grinding compartments for wear of their internal fittings. Furthermore, in order to reduce downtime and wages, the necessary preparations should be made in advance, i. e., before actual plant shutdown. These include taking samples of the material before and after the mill when the grinding plant is operating under steady-state conditions. For process engineering checks it is instructive to include the performance of equipment "upstream" and "downstream" of the mill in the assessment of the functioning of the mill itself.
The personnel who have to be in attendance for opening and entering the mill should be summoned in good time, and the necessary tools, ladders, lamps and any measuring instruments that may be required should be in readiness. Samples of material at points spaced 1 m apart should be taken from within the mill and stored in identifiable containers. Properly trained and experienced personnel should preferably be used for this work in order to reduce the risk of mistakes in sampling and measurement. Obviously, it is necessary to take adequate safety precautions so as to ensure that the mill will not be inadvertently started while there are men inside it. A supervisor should be present, outside the mill, while the internal inspection is being made.

3.7 1 Mechanical checks

Despite the use of high-grade engineering materials and fixing techniques in modern mills, damage to internal fittings cannot be ruled out. As present-day grinding plants are in many instances operated under remote control – i. e., they

are started, stopped and monitored from control stations some considerable distance away – there is a risk that relatively minor initial damage may produce major consequential damage before it is detected.
It is advisable to take every available opportunity to detect and remedy possible sources of trouble in their early stages. In carrying out the inspection of the mill it is therefore important also carefully to examine the internal fittings.

3.7.2 Mill lining

As a rule, the interior of a tube mill is accessible only through manholes. These can most conveniently be opened and closed when they are positioned at the top of the shell when the mill has stopped. However, if this is adopted as standard practice, it means that always only the same approximately two-thirds portion of the circumference not covered by the grinding media can be inspected. In order to examine the other parts of the lining, the inspection should from time to time also be carried out with the manholes in a different position. The task of handling the heavy manhole covers under such conditions can be facilitated by replacing the covers by temporary lightweight ones, e. g., consisting of 10 mm thick steel plate, when the manholes are in the top position.
Fractured or broken parts of the lining are a potential source of trouble. The liners thus affected should be replaced by new ones, even if the fractured pieces appear to be firmly interlocked and perhaps also held in position by grinding media wedged into the lining.
If a fairly large number of liners are found to be damaged in a particular area of the mill, it is necessary not only to repair the damage, but also to find the cause. Besides the more obvious possible causes of damage, such as material flaws or insecure fixing of liners, there are others, including: loading percentage too low, so that cataracting grinding media overshoot the bed and strike the liners; too low a rate of feed under continuous operating conditions (under-loading of the mill); too coarsely graded grinding media charge; incorrect interadjustment of the hardness values of the grinding media and liners.
For operational reliability of the lining the liners must be satisfactorily supported and secured. If some or all of the liners are bolted, the bolts should be checked from time to time in order to ascertain that they are still tightened to the correct torque recommended by the manufacturer.
Already at the time of first installing the lining it should be ensured that the bearing surfaces to which the liners are fixed are properly even. Any irregularities such as burrs or fins must be removed before the liners are fixed.

3.7.3 Intermediate and discharge diaphragms

The diaphragms (division plates or partitions) are subjected not only to wear, but also to considerable cyclic mechanical loads. These affect more particularly the supporting frames on which the liner plates or the slotted screen plates of the diaphragms are mounted. The inevitable differential movements between these frames and the mill shell, and also between them and the plates they carry, have to

be resisted by the fixing bolts. It is these bolts in particular that are liable to fracture.
Having regard to the cost and effort of making good the consequences of damage to, say, an intermediate diaphragm, it is reasonable and advisable to test each and every bolt by tapping it with a hammer.
The liners or the slotted screen plates of mill diaphragms are often secured by means of shear bolts. Particularly in mills of relatively large diameter it is difficult, if not indeed impossible, to replace individual bolts near the periphery of the diaphragms. If the attachment of a liner or screen plate appears to be critically weakened, it may be advisible, as an interim measure till the next major overhaul, to remove the sector affected and substitute an ordinary steel plate cut to the appropriate shape. To enable this temporary sector to be secured by bolting, an opening should be provided in it through which it is possible to reach the back of the plate and manipulate the fixing bolts. When the plate has been fitted to the diaphragm, the opening should be closed with the piece of steel originally cut away to form it. This piece should then be welded in position.
Replacement of the bolts for fixing the supporting frame to the shell in a large mill, which are likewise difficult to get at, can be facilitated by inserting a piece of wire from outside the mill through the hole in the shell and welding the new bolt to the end of the wire inside the mill. The bolt can then be pulled carefully between the lifters into its hole.
In inspecting the diaphragms in the mill all their visible parts and those of the supporting frames should be checked for the presence of cracks. The condition of the screens or perforated plates in the middle of the diaphragms should also receive adequate attention. Any metallic foreign bodies that have become wedged in the slots of diaphragms and protrude into the grinding compartment should be removed because impact with large grinding media may produce a wrenching effect that will fracture the bars adjacent to the slots.

3.7.4 Feed and discharge equipment

The feed and discharge devices are frequently provided with lifting and/or conveying inserts. As these internal fittings are also subject to considerable wear, they should be inspected at suitable intervals. Their fixings should be checked and, if necessary, renewed.

3.7.5 Other checks

During fairly long shutdown periods the critical or especially severely stressed parts outside the actual grinding compartment of the mill should also be duly inspected. The following are especially important:

Trunnion bearings

The trunnion bearings on many mills are lubricated by means of oiling rings and wipers. The condition of these components should be checked. In particular, depending on the design features in any given case, the joints of the oiling rings should receive attention. Worn wiper elements should be renewed in good time. The trunnions themselves should be examined for the presence of scratches and grooves.

To ensure operational reliability of the trunnion bearings it is essential not only to supply them adequately with lubricant, but also to ensure that the lubricant is free from contamination. The bearing housing should be effectively sealed. The sealing elements, usually rubber lip seals or fabric seals, should be adjusted or renewed, as necessary. It should also be ensured that these sealing elements are coated with a film of lubricant to protect them against wear.

Drive

In the case of mills equipped with girth gear and pinion drive the tooth bearing, the condition of the tooth flanks and the lubricant film should be checked at regular intervals. If spray lubrication is employed, the spraying devices should likewise be regularly checked to make sure that they are functioning properly. This can be done by laying a sheet of paper on the part of the girth gear destined to receive the atomized spray and to allow the lubricating system to perform one operating cycle.
With properly functioning spray nozzles the lubricant should be uniformly distributed over the full width of the girth gear. The quantity of lubricant dispensed in each successive spraying operation can be determined from the difference in weight obtained by weighing the sheet of paper before and after spraying. The result should, with due regard to the number of spraying operations that the lubricating system performs per hour, be checked against the recommendations of the mill manufacturer or lubricant supplier.
The requirements applicable to the seals of the girth gear housing are similar in principle to those already stated for the trunnion bearing seals. Here, too, it is important to prevent dust getting into the housing.

Mill heads

Although the mill heads (end walls) are designed on the basis of sound structural and metallurgical principles, and are manufactured and tested with all possible care, fracturing and damage cannot be completely ruled out. It is therefore advisable also to inspect these components at regular intervals, with particular attention to the transition between the trunnion and the head. To enable any cracking to be detected as early as possible, it is desirable to keep these parts free from dust.

Mill shell

The same as has been said concerning the mill heads applies also to the shell, i.e., the cylindrical body of the mill. The parts especially at risk are those where, as a result of the unavoidable deformation and deflection of the shell, stress concentrations are liable to occur during operation. Such parts are the joints of the cylinder segments and those of the manhole strengthening surrounds, which are usually welded to the shell. These areas of the mill shell should therefore also be inspected.

3.8 Process engineering checks

3.8.1 Determining the loading percentage

The loading percentage, or filling ratio, is defined as the ratio of the bulk volume of the grinding media to the total internal volume of the grinding compartment. For practical purposes it can be expressed as a ratio of cross-sectional areas:

$$f = \frac{\text{cross-sectional area of grinding media charge}}{\text{internal cross-sectional area of mill}} = \frac{F}{D_{Li}^2 \times \pi/4}$$

For determining the ratio, the internal diameter of the mill and the distance from the top surface of the grinding media bed to the highest point of the mill lining should be measured. If the lining is provided with profiled, e. g., corrugated or stepped, liners a suitable correction should be made and the average diameter be adopted (Fig. 38).

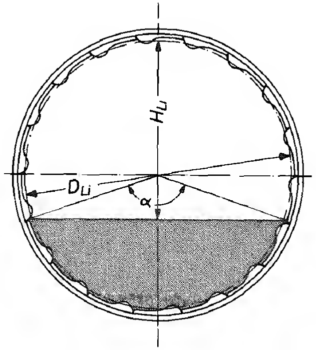


Fig. 38: Average internal dimensions with profiled liners

Notation:

D_{Li} = internal diameter of the mill (within liners) [m]
H_{Li} = distance from top of grinding media bed to highest point of the lining [m]
r = internal radius = $D_{Li}/2$ [m]
α = central angle [degrees].

The central angle is determined from: $\cos \alpha/2 = 2\,H_{Li}/D_{Li} - 1$.

The filling ratio is: $f = \left(\frac{\alpha}{360} - \frac{\sin \alpha}{2\pi}\right)$.

Alternatively, the following approximation may be adopted $f = 1.068 - 1.164\,H_{Li}/D_{Li}$.
An even simpler approximation is obtained by counting the number of exposed liners visible around the circumference, i. e., not covered by grinding media, and relating this to the total circumferential number of liners. This yields the formula:

$$f = 1.34 - \frac{0.172 \times \text{number of exposed liners}}{D_{Li}}.$$

For the sake of better accuracy, the dimensions D_{Li} and H_{Li} should be determined as averages from a number of measurements, especially if the mill is fitted with profiled liners.
The filling ratio is often expressed in per cent, and as such is more particularly known as the loading percentage or per cent loading of the mill.
The weight of the grinding media charge of the mill, or of a compartment thereof, can now be calculated from

$$G = \frac{D_{Li}{}^2 \times \pi}{4} \times f \times q_{GM} \times L_M \ [t].$$

where L_M is the effective length of the mill or compartment [m] and q_{GM} the bulk density of the grinding media [t/m^3]. The bulk density for steel grinding media (balls or Cylpebs) ranges from about 4500 to 4800 t/m^3. For normal grinding media mixtures an average of 4550 t/m^3 is generally reasonable.

3.8.2 Grinding media classification

For effective size reduction there should be an appropriate ratio between the size of the feed material particles and the mass of the individual grinding media. As the size of the particles decreases along the mill, the mass and therefore the size of the media (ball diameters, etc.) should correspondingly decrease. This condition can be satisfied by providing the mill with so-called classifying liners.
In order to obtain an objective assessment of the effectiveness of the grinding media classification and to determine what changes occur in the grinding media grading over fairly long periods, it is advisable to perform checks from time to time. For that purpose, samples of the media are taken from the top layer at points spaced at equal distances along the mill. The number of media to be sampled at each point should either be determined in advance or should be the total number found to be present within a specified area of the top layer. The samples thus obtained are weighed and the average weight and (in the case of balls) the corresponding diameter are calculated. Obviously, the number of grinding media taken at each sampling point should be sufficient to enable reliable averages to be determined. The guarantees issued by the suppliers of classifying liners are often based on the averages of 100 balls or other grinding media. For routine checks in the works, however, smaller numbers – 50, for example – will yield sufficiently accurate results.

3.8.3 Determining the number of fractured grinding media

Despite reliable production methods and regular quality control, defects of manufacture in grinding media cannot be ruled out. In the mill such defects or flaws may result in spalling or fracturing of the media beyond an acceptable limit. The fragments detached from them are liable to have an adverse effect on grinding performance.
The proportion of fractured grinding media in the whole charge can be estimated by a sampling method similar to that used for monitoring the grinding media classification. The grinding media and fragments thereof which are present within a predetermined circular area in the top layer are weighed and sorted. The fractured proportion is expressed as a percentage of the total weight of the sample. Fragments of larger grinding media are classified in the mill as though they were small grinding media, so that a higher proportion of fragments is bound to occur in the samples obtained close to the outlet end of the grinding compartment. The samples should preferably be taken at regularly spaced points (1 m apart, say) along the length of the mill.
The proportion of damaged grinding media is expressed by.

$$\text{fractured percentage} = \frac{Q_{D1} + Q_{D2} + Q_{D3} + \ldots Q_{Dn}}{Q_{S1} + Q_{S2} + Q_{S3} + \ \ .\, Q_{Sn}} \times 100\ [\%],$$

where Q_D is the weight of the damaged proportion in an individual sample and Q_S is the weight of an individual sample.

3.8.4 Checking the lining

The design and configuration of the mill lining is of major influence on the motion of the grinding media charge and thus on the comminuting action developed by it. Wear that reduces the profiling of the liners, so that their lifting action is impaired, will promote undesirable premature sliding back of the grinding media. As a result, the point of detachment of the media from the wall of the mill is gradually shifted lower down. The power consumption, and therefore the energy available for size reduction, diminish in consequence. For this reason it is necessary to inspect the condition of the lining from time to time.
Wear of the corrugations, ridges or other features of the lining can be checked with the aid of templates conforming to the profiling of the lining in its original (new) condition. By applying a template to, for example, liners that can be conveniently reached from a manhole, changes in the condition of the lining can quickly be detected.

3.8.5 Checking the diaphragms

The purpose of the diaphragms with their slotted plates is to act as screens which allow feed material which has been sufficiently reduced in size to pass to the next grinding compartment or to the mill outlet, while grinding media and oversize particles are retained. The effective cross-sectional area of the openings in the

diaphragms should be sufficiently large to enable the fine particles as well as the air or hot gas (for drying the material in the mill) to pass at the required rate. Fragments from fractured grinding media, heavily worn media or the feed material itself – especially if it is too moist and/or the air flow through the mill is inadequate – may cause choking of the slots in the diaphragms and thus obstruct transfer or discharge of the material.
To reduce the risk of choking, the slots are so formed that they widen in the direction of passage of the material through them. With increasing wear the slots become wider and thus let coarser particles through This oversize material is liable to cause problems in the fine grinding compartment.

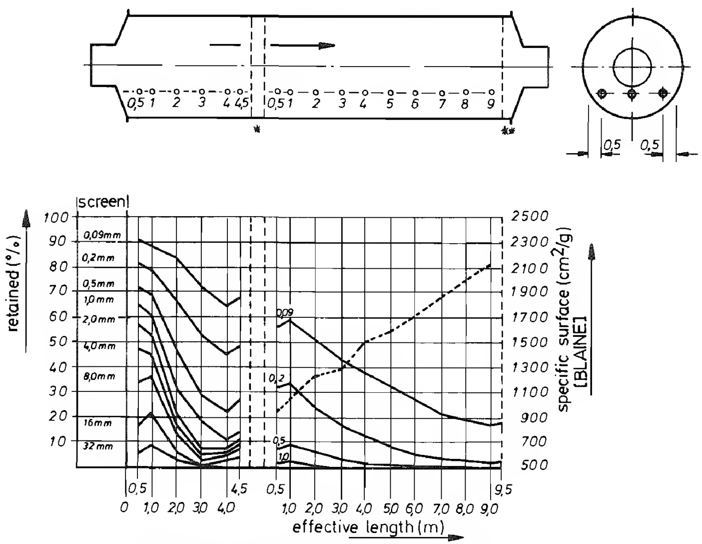
3.8.6 Checks in the interior of the mill

For the checks and inspections described here it is important that the grinding plant should be shut down direct from steady-state operation with its normal throughput, without any alterations – either before or after shutdown – that may affect the granulometric composition and quantity of feed material inside the mill. This requirement can perhaps most readily be fulfilled by stopping the mill quickly by means of the emergency switch or, in the case of a fully interlocked system, by switching off an important unit of plant downstream of the mill. The mill fan should also be stopped at the same time, otherwise the air sweeping through the mill may alter the condition of the bed of material and thus cause incorrect conclusions to be drawn.
High temperature in the mill may, however, make it necessary to cool the interior before it can be entered for inspection. In that case the fan will have to be switched on again, but taking care that it is started with its control damper or inlet vanes closed and that these are subsequently opened up only to such an extent as is necessary to lower the temperature sufficiently.

3.9 Size reduction progress

For monitoring the size reduction progress, i. e., the degree of comminution of the feed material achieved on its way through the mill, samples of material should be taken at points spaced 1 m apart along the mill, starting at a distance of 0.5 m from the mill inlet or the intermediate diaphragm.
As the granulometric composition of these regularly spaced samples is likely to vary according to whether the sample is taken at one particular spot or comprises several samples taken across the width of the bed of material, it is advisable to adopt an agreed sampling procedure before carrying out the checks. It is recommended that each sample at 1 m intervals should itself be composed of three individual samples consisting of equal volumes of material. Two of these samples should be taken at a distance of about 0.5 m from the lining on each side, and the third sample from the middle of the bed. The last sample along the mill should be taken at 0.5 m before the intermediate diaphragm or discharge diaphragm. Because the bed of material falls away here, it is often difficult to obtain samples at such

points. Yet it is these samples that are particularly informative, and it is therefore worth making the effort to remove some layers of grinding media in order to reach the material. The presence or absence of a high concentration of coarse particles of material in this part of the mill, i.e., close to the intermediate diaphragm or the outlet, can provide important information on the condition and effectiveness of the grinding media charge.
The samples thus obtained at 1 m intervals along the mill are screened and the cumulative quantities retained on the screens are plotted as a curve in a diagram. The ordinates represent the cumulative percentages (by weight) retained, while the distances in metres along the mill are marked on the horizontal axis. Points of equal particle size in the diagram are connected to one another. In addition, with appropriate feed material and fineness of grinding, the specific surface values may also be determined and be plotted. The "grinding diagram" obtained in this way gives clear information on the quality of the size reduction process (Fig. 39).



* intermediate diaphragm 8 mm width of slots
* * discharge diaphragm 10 mm width of slots
—— retained
---- specific surface

Fig. 39: Grinding diagram of a clinker grinding mill (closed circuit operation with bucket elevator)

3.9.1 Height and condition of the material bed

The bed of feed material being ground should cover the top layer of grinding media, but not to any appreciable depth. If the bed is too high or too low, it indicates defective composition of the grinding media charge. With too coarse media the bed will be too low, and vice versa.
Wavy or hump-like irregularities in the bed of material and grinding media may be caused by varying resistance in the bed. More particularly such variations may be due to: transition from non-classifying to classifying liners, unfavourable grinding media grading (with too abrupt a transition from coarse to fine media), inadequate initial comminution in the first few metres of the mill and therefore too much coarse material arriving in the zone with finer grinding media.
In the vicinity of the intermediate and discharge diaphragms there should be a distinct falling-away of the bed of material. If this is not the case, or if indeed there is a local accumulation of material instead, this is generally attributable to inadequate discharge capacity of the diaphragm, i.e., the effective area of its slots is inadequate. In mills operating in closed circuit with a classifier a distinct accumulation of feed material often occurs near the mill inlet, which is due to tailings from the classifier continuing to enter the mill for some time after the plant has been stopped. It is not, therefore, an indication of inadequate comminuting action.
To enable the condition inside the mill to be assessed, the height of the material bed and the appearance presented by the media and material should likewise be measured and recorded at each point where samples are taken for establishing the grinding diagram. The material bed heights may be included in the diagram. Quite often, distinct functional relationships are seen to exist between the cumulative screen curves, the curve for specific surface and the depth of the bed of material.
If the bed is fairly high, the measurement can be performed by inserting a strip of cardboard or sheet metal into it. The strip should have a width equal to at least twice the largest grinding media size.

3.9.2 Build-up of material on liners and grinding media

Build-up (caking of material) on liners and grinding media may be due to various causes. Their interactions giving rise to this undesirable phenomenon have not yet been fully explained.
The following causal factors may be mentioned:
- static electric charging and free surface energy,
- adsorption;
- mechanical pressure.

Higher temperatures in the mill increase the tendency, which also appears to increase not only if there is too much moisture in the mill atmosphere, but also if the atmosphere is too dry. As the caked material has a cushioning effect which impairs the grinding performance, it is likewise something to look out for during an internal inspection of the mill. The nature and extent of any build-up should be noted. The information thus collected should be included in the mill record sheets. More particularly, the following data should be obtained:

- where the first build-up occurs (at what distance from the mill inlet or intermediate diaphragm);
- what parts are affected (liners, grinding media);
- extent of the build-up (approximate estimate of the areas covered with caked material on that part of the lining which is visible);
- strength and thickness of the build-up material (e. g., can be easily wiped off, or cannot be removed without the aid of a tool).

Because of the many causes and their interaction it is not possible to lay down generally-valid rules for the prevention of build-up. The measures to be taken must therefore be decided for each individual case. Quite often, however, the problem can be overcome by improving the air flow conditions in the mill. For example, if it is confined to the mill lining, it may be due merely to condensation of moisture. Improved air flow may provide the remedy. Other possible measures to combat build-up are: injection of water into the mill in order to lower the temperature and/or the addition of a grinding aid to neutralize the forces associated with free surface energy.

3.9.3 Determination of wear

Besides the process engineering consequences of wear, such as decline in grinding performance and undesirable changes in the granulometric composition of the product, the economic aspects of wear are also of importance in connection with the operation of grinding plants.
In order to obtain precise information on wear and be able to compare the behaviour of parts made from different materials and/or supplied by different manufacturers, it is advisable to observe and record the wear of the grinding media, liners, intermediate and discharge diaphragms.
Reliable information on specific rates of wear moreover facilitates the spare parts inventory and the planning of repairs.

3.9.3.1 Grinding media wear

Weighing the whole grinding media charge before putting it into the mill and subsequently – after a fairly long period of service – weighing it again is certainly the most accurate method of determining the rate of wear. However, because of the considerable effort it involves, it is a method which, if at all, can be considered only for very small mills (e. g., experimental mills).

The specific wear is. $Q_{spec} = \dfrac{\Delta Q_{media}}{\Delta Q_{material}}$ [g/t],

where ΔQ_{media} is the difference in weight of the grinding media (in grams) before and after the period of service, and $\Delta Q_{material}$ is the quantity of feed material that has been put through the mill during that period (in tonnes).
As an alternative to this laborious and therefore rather impracticable method, the specific wear of particular grinding media sizes or size fractions can be estimated by

Table

Grinding media: balls (density 7.7 g/cm³)

diameter mm	surface area cm²	volume cm³	weight g	surface area/t m²/t	number of balls/t
120	452	905	6964	6,5	143
110	380	697	5364	7,1	186
100	314	524	4032	7,8	248
90	254	382	2938	8,7	340
80	201	268	2063	9,7	484
70	154	180	1382	11,1	723
60	113	113	870	13,0	1150
50	78,5	65,5	504	15,6	2000
40	50,3	33,5	258	19,5	3900
30	28,3	14,1	109	26,0	9200
25	19,6	8,18	63,0	31,2	15900
24	18,1	7,24	35,7	32,5	17900
23	16,6	6,37	49,0	33,9	20400
22	15,2	5,58	42,9	35,4	23300
21	13,9	4,85	37,3	37,1	26800
20	12,6	4,19	32,2	39,0	31000
19	11,3	3,59	27,6	41,0	36200
18	10,2	3,05	23,5	43,3	42500
17	9,08	2,57	19,8	45,8	50500
16	8,04	2,14	16,5	48,7	60600
15	7,07	1,77	13,6	52,0	73500
14	6,16	1,44	11,11	55,7	90400
13	5,31	1,15	8,9	59,9	112900
12	4,52	0,91	7,0	64,9	143500

weighing samples containing representative numbers of these media. In many cases it is just these specific wear rates that are of interest to the mill operator. It is a suitable method when starting with a grinding media charge consisting of completely new media or otherwise of very carefully selected and graded media which are all of the same quality.
Before commissioning the mill with a newly assembled grinding media charge, a number of media of each size are taken and weighed, in order to determine the average weight of one ball, Cylpebs, etc. of that size. The number of media to be taken in each sample will depend on how greatly the individual weights vary within a certain nominal size and on the degree of accuracy required. For ordinary works investigations it will normally be sufficient to take 30 grinding media of each size.
After a suitably long period of service in the mill, the same numbers of the individual sizes are again taken and the average individual weights determined. The wear that has occurred is obtained by determining the difference between the original weight (new media) and the weight after service and multiplying this by the total number of media, of each size, with which the mill was charged. This method becomes impracticable when wear has progressed to such an extent that it is no longer possible reliably to determine the original nominal sizes of the grinding media.
If grinding media of a different quality from the existing charge are added with a view to investigating their wear behaviour, and if these new media do not differ substantially in shape and dimensions from the existing ones, they should be provided with identification marks (grooves or drilled holes) to enable them to be identified from the others after a period of service in the mill.
A different procedure consists in determining the filling ratios before (f_1) and after (f_2) a sufficiently long period of service. The weight calculated from the difference in filling ratio (loading percentage) provides an indication of the wear that has taken place. It should be borne in mind, however, that the average bulk density of the grinding media mixture will undergo a change in consequence of the different wear rates of the respective grinding media sizes. It should in each particular case, having regard to the desired accuracy, be considered whether or not a correction to take account of this change in bulk density is necessary.

The wear is expressed by: $\Delta Q = \frac{D_{Li}^2 \; \pi}{4} \times L_{eff} \times \Delta f \times q_b$ [t],

where:

Δf = difference in filling ratio before and after the service period considered $= f_1 - f_2$
ΔQ = grinding media quantity lost by wear [t]
D_{Li} = internal diameter of mill [m]
L_{eff} = effective length of mill or grinding compartment [m]
q_b = average bulk density of grinding media charge [t/m³].

Specific wear: $Q_{spec} = \Delta Q \times 10^4 / Q_{material}$,

where $Q_{material}$ is the throughput of feed material during the service period considered (in tonnes).

3.9.3.2 Lining wear

Wear of the mill lining can impair its purely protective function of preventing damage of the mill shell and moreover diminish its effectiveness in lifting and classifying the grinding media. For process engineering as well as economic reasons it is therefore necessary to monitor the wear behaviour of the lining.
The most reliable method of quantifying the wear is to remove some liners, from points uniformly distributed along the length of the mill, from time to time and compare their weight with the weight of those plates in the new condition. As this is a very laborious and time-consuming procedure, however, in practice a somewhat less accurate but more convenient method will generally be adopted.
One such method is based on measuring the internal diameter of the mill, i. e., within the lining, applying a correction to allow for the average profile depth on corrugated or stepped liners. The volumetric amount of lining wear can be calculated from the difference between the diameter of the worn lining and that of the lining in its new condition. The specific wear is:

$$Q_{spec} = \frac{V_{wear} \times q_{lining}}{Q_{material}} \text{ [g/t]},$$

where:

V_{wear} = volumetric wear of the lining [cm³]
q_{lining} = specific gravity of lining [g/cm³]
$Q_{material}$ = throughput of feed material during the service period considered [t].

A drawback of this method is that, to obtain reliable results, the measurements must be performed very accurately and that changes in the profile of the liners due to wear are very difficult to take into account.
Another method consists in comparing the liners with templates corresponding to their profiles in the new condition. After appropriate service intervals these templates are applied always to liners at the same points in the mill, e. g., at joints between diaphragm plates, or between end wall liners, or at shell liners that can be reached from a manhole. The volumetric wear can be determined from the difference between the template profile and the profile of the liner in its actual (worn) condition. If the specific gravity of the lining material and the number of liners are known, the weight of this material lost by wear can be approximately calculated. Taking account of the total throughput of feed during the period considered, the specific wear can then be found.

3.9.3.3 Wear of the diaphragms

The diaphragms in tube mills are subject to considerable wear from the grinding media rolling, cascading and cataracting in contact with them. Determining the actual pattern of wear for calculating the loss of lining material from measured differences in volume is usually very laborious. For practical purposes, however, it will usually be sufficient regularly to determine the thickness of the plates at the most heavily worn points and estimate the service life from the measurements. These can be facilitated by using a piece of wire bent at right angles at one end, which is inserted through a slot in the diaphragm and turned. With closed rear wall plates of diaphragms the thickness measurements can be performed at the joints. When the diaphragm plates are due for renewal, it is advisable to take the opportunity to determine the actual rate of wear by comparing the residual weight with the weight of the plates as they were when new.

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4 Roller mills

This class of mills comprises many variants which nevertheless have certain basic features in common. There is some confusion in the terminology designating these mills, especially in the German language. In English, "roller mills" has come to be widely accepted as a generic term including even those machines in which the rollers are in fact balls. Designations such as ring-roller mills, ring-ball mills, bowl mills, etc. are generally confined to the description of specific types. All these machines are characterized in having rollers (or comparable other grinding elements) which travel in a horizontal circular path on a bed of feed material with which they are pressed in contact by vertical forces applied externally to them, the material being comminuted by a combination of compressive and shearing action.

Roller mills employed in the cement industry have grinding elements of various

shapes. Thus, in some mills they are cylindrical rollers, in others the rollers are of truncated-conical shape or have flat lateral faces and a convex circumferential surface. Some leading manufacturers equip their mills with balls as the grinding elements. The force that keeps the rollers or balls pressed in contact with the bed of material on the grinding path may be exerted by gravity, centrifugal force, spring pressure, hydropneumatic action, etc.

In recent years, roller mills ranging up to very large throughput capacities have come into widespread use for raw material and coal grinding in the cement industry. Technical development has reached an advanced stage, but has by no means been completed, and there are as yet no discernible reasons why even bigger mills with higher throughputs should not be introduced. There also exist interesting prospects for using these machines as finish grinding mills, i. e., for clinker grinding. Encouraging results have been obtained in this direction, but it still remains to be seen whether economical solutions will emerge for the major problem of wear and the associated effects on the quality of the cement produced by grinding in roller mills.

On the other hand, roller mills have already long established themselves as very suitable for coal grinding, i. e., for the production of pulverized fuel (see Section 5.5.2). The widespread return to pulverized coal and lignite in cement manufacture is having a stimulating effect on the development and optimization of these mills which, for this type of work, are usually of relatively small size and operated with direct firing systems.

4.1 Roller mill design features

In view of the many different manufacturers and design variants, both in Germany and in other countries, it is obviously not possible to deal with all the various makes of roller mill in this book. It will, however, be endeavoured to classify and briefly describe the familiar main types with reference to the mills supplied by some manufacturers mentioned by name, on the understanding that this must not be construed as implying preference in terms of performance or quality.

A common characteristic of all the mills described here is that size reduction is effected by rollers or comparable grinding elements travelling over a circular bed of material and that the material, after passing under the rollers, is subjected to a preliminary classifying action by a stream of air sweeping through the mill. Depending on the air flow velocity, a certain proportion of the pulverized material is thus carried into a classifier (air separator) which normally forms an integral feature of the upper part of the casing of the mill. Oversize particles rejected by the classifier fall back into the grinding chamber, while the fines are swept with the air out of the mill and are collected in a filter or a set of cyclones. As the pneumatic conveying of the material in the mill to the separator requires considerable air flow rates, and as the material leaving the grinding bed and carried up into the classifier comes into intimate contact with the air, roller mills are especially suitable for the drying of moist feed material in combination with grinding. This is particularly advantageous because these mills can accept large quantities of hot air or gas at relatively low temperatures such as commonly occur in the waste gases of cement manufacturing plants.

4.1.1 Mills with truncated-conical rollers (Loesche mills)

Two or more conically tapered grinding rollers in fixed mountings travel on an annular path on the upper surface the revolving grinding table on which the bed of feed material lies. The rollers are mounted on swivel arms on which they can be swung out for repairs or maintenance. Roller pressure is exerted by springs on smaller machines and hydropneumatically on larger ones (Fig. 40). The table on which the renewable liner segments of the grinding ring forming the roller path are mounted is driven through gears in a gearbox which is designed to resist the pressure exerted by the rollers. The material to be ground is fed centrally onto the grinding table and is carried by centrifugal force, due to the rotation of the table, to the roller path. At the circumference of the table is a raised rim, a so-called dam ring, by means of which the depth of the bed of material can be adjusted. Between the outer edge of the table and the casing of the mill is a stationary ring comprising ports through which air is admitted from under the grinding table into the grinding and classifying chamber.
The pulverized material that spills over the rim is caught by the upward stream of air issuing from the ported air ring. The air is guided and accelerated by vanes or louvres, so that a kind of fluidized bed is formed. Widening of the flow cross-section causes the air velocity to decrease over the rollers, so that coarser particles

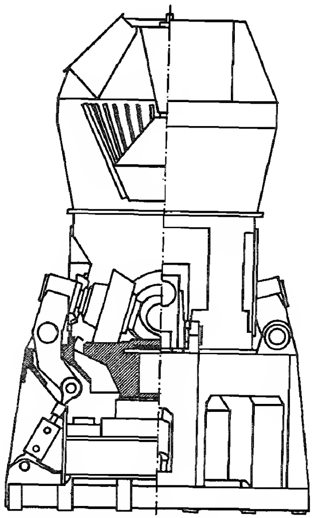

Fig. 40: Mill with truncated-conical rollers (Loesche GmbH)

fall back onto the table. The particles swept up to the rotor-type classifier undergo a separating action, the oversize fraction likewise falling back onto the table for further grinding, while the fine particles (the finished product) are carried out of the mill.
Depending on the grindability of the material and the air flow rate, a recycling of the material thus develops inside the mill casing. The circulating load may amount to as much as 8 to 10 times the rate of fresh feed to the mill. This recycling requires a high air flow rate a fact which, as already stated, can be turned to advantage for drying the material during the grinding process. It is thus possible to grind and dry cement raw materials with up to 18% moisture content without unacceptably lowering the throughput of material. For coal grinding it is possible even to accept a feed moisture content ranging up to 25%.
The bladed rotor classifier mounted over the grinding chamber has variable speed control. In rotates on a vertical axis and its rotary motion imparts a horizontal centrifugal acceleration to the mixture of air and material particles rising from below. The oversize particles, on account of their greater mass, are deflected further out of the air stream, impinge on the wall of the casing and fall back into the grinding chamber. The fines discharged from this classifier are characterized by about 1% retained on the 0.2 mm sieve and 12% on the 0.09 mm sieve.
A notable feature of the mill described here is that its rollers are mounted in bearings that are outside the grinding chamber with its high dust concentrations and elevated temperature.

4.1.2 Mills with convex-surfaced rollers (Pfeiffer MPS mill)

In principle this roller mill is similar to the machine described in the preceding section. It is equipped with three rollers, likewise in stationary mountings, running on an annular path of concave cross-sectional shape to accommodate the convex surfaces of the rollers. The material is fed from one side onto the rotating grinding ring. The grinding pressure is developed by the dead weight of the rollers operating in conjunction with a hydropneumatically tensioned spring system. After being discharged from the edge of the grinding ring the pulverized material is entrained by the upward stream of air issuing from the ported air ring and undergoes preliminary classification in the same way as in the Loesche mill.
Oblique setting of the ports imparts a circulatory motion to the material in the direction of rotation of the rollers. The coarse particles that fall back onto the grinding ring here and the oversize rejects from the classifier are returned to the roller path to undergo further size reduction (Fig. 41), while the fines are carried with the air stream out of the top of the mill and classifier casing. The cut size of the rotor classifier is adjustable.
In terms of size reduction performance the MPS mill is similar to the Loesche mill of comparable specification, but its very ample flow cross-sections in the grinding chamber allow even larger air flow rates through the mill. According to information supplied by the manufacturer, cement raw material with above 20% moisture content can be dried in the mill to below 1% residual moisture.

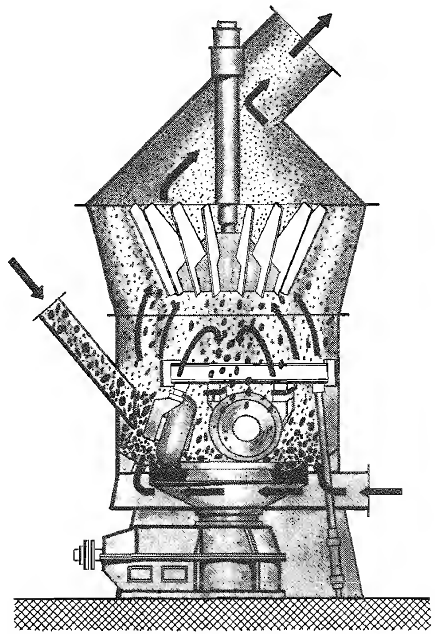

Fig. 41: Mill with convex-surfaced rollers (Gebr. Pfeiffer)

4.1.3 Mills with spherical grinding elements (Peters mill)

In this type of mill, known also as a ring-ball mill, the grinding action is performed by balls set close together and rolling on a power-driven rotating grinding ring. At the top the balls are held in position and pressed down – by springs or hydro-pneumatically – by a pressure ring, which is stationary. The whole assembly resembles a very large ball bearing.
The material is fed centrally on to the grinding table and carried by centrifugal force to the grinding ring on which it is pulverized by the balls rolling over it. At the perimeter of the ring the pulverized material is entrained in an upward stream of air and undergoes preliminary classification, as in the previously described mills, after which it passes to the classifier (usually of the static type), where the oversize material is rejected and falls back into the mill. The fines are carried out of the mill in the air stream (Fig. 41 a).
By passing hot air or gas through the mill, drying performance comparable to that of the other roller mills can be obtained.

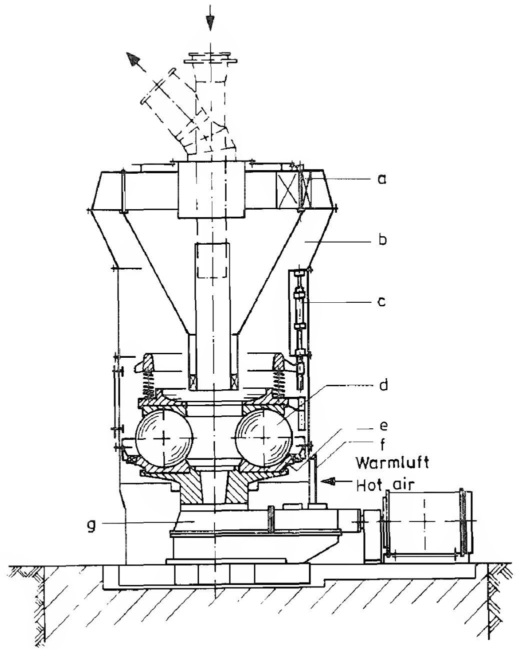

Fig. 41 a: Mill with spherical grinding elements (Claudius Peters AG)

4.2 Grinding action developed in roller mills

The material is comminuted by the grinding elements rolling on a circular bed of feed material. The larger pieces of material are crushed by the rollers as in a roll crusher, while the smaller ones are reduced by rubbing action. The pulverized material spilling over the edge of the grinding table or grinding bowl – the terminology tends to vary from one manufacturer and mill design to another – is entrained by a high-velocity stream of air, so that the smaller particles are swept upwards into the classifier and the coarser ones fall back onto the roller path. This is the preliminary classifying effect, as distinct from the final separation accomplished in the internal classifier in the upper part of the casing.
Because of the short residence time of the feed material in the grinding chamber as compared with that in a tube mill, the bed of material is kept substantially free from fine particles which do not require further grinding, unnecessarily load the mill and

tend to form undesirable agglomerations. The important basic conditions for effective grinding in a roller mill are that the grinding elements develop a good draw-in action and adequate pressure and that a stable bed of material is formed.

4.2.1 Draw-in action of the grinding elements

As in a roll crusher, there is a geometric relationship between the diameter of the grinding elements (rollers or balls) and the maximum particle dimensions that the mill can accept. In roller mills, maximum feed particle sizes of between about 1/20 and 1/15 of the roller (or ball) diameter are permissible. If material coarser than this is fed to the mill, there is the danger that the coarse particles will not be drawn in under the rollers but will simply be displaced, i. e., pushed along in front of them.
Furthermore, within the permissible maximum particle size limit, the draw-in action is governed by the granulometric composition and coefficient of friction of the feed material. Thus, the bed of material should possess adequate stability so as not to be displaced by the rollers. Also, in order that the rollers do indeed roll on the material and not merely slide along, a sufficiently large frictional force must be developed between their circumference and the material.
It may occur that, while the mill is operating under steady-state conditions, the granulometric composition of the feed material changes drastically, e. g., due to

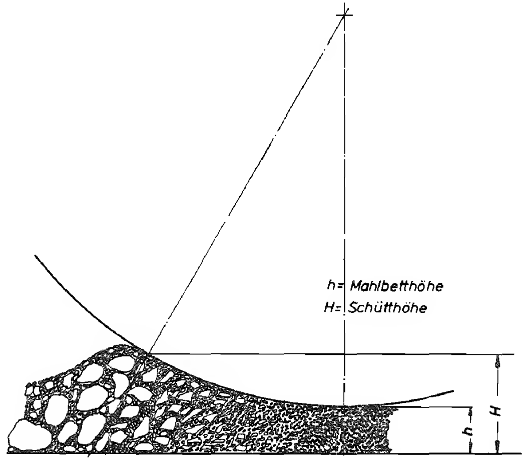

Fig. 42: Draw-in action of feed material between roller and grinding ring
h = depth of bed
H = initial depth

segregation on emptying the feed hopper, so that the mill temporarily receives only fine material. This may adversely affect the stability of the bed: part of the material is displaced, the depth of the bed is therefore reduced, and (assuming the pressure on the rollers to be unchanged) the specific pressure exerted on the material is increased. It may thus occur that the rollers "punch through" the bed in places, resulting in "bumpy" running.
As the condition of the feed material is liable to vary with regard to its grindability, composition, granulometry and moisture content, mill designers strive to achieve adequate draw-in capacity of the rollers that will cope with any variations likely to occur in the feed material. Measures to achieve this include: providing the rollers and roller path with raised profiling (ridges) and utilizing the joints of the renewable segments on these components to provide positive grip. Another possibility is to use alternate segments with different wear properties or to form ridge-type raised features on the rollers by means of highly wear-resistant weld metal deposited with special electrodes.
A dam ring at the perimeter of the grinding ring serves to maintain the required stability and depth of the bed of material. Furthermore, in large machines with hydropneumatically applied grinding pressure, the pressure can be varied to suit the existing conditions of grinding.

4.2.2 Grinding action

The grinding that the material undergoes between the rollers and the roller path on the grinding ring comprises the following actions:

- **Draw-in of the material:** The particles of feed material are gripped between the roller and the grinding ring. The larger ones, which project above the others and are the first to be subjected to the crushing action, are broken down. This size reduction is of course promoted by the fact that the pressure is initially concentrated on these larger particles, so that their compressive strength is quickly and greatly exceeded. The pressure exerted by the roller is then transferred mainly to the particles ranking next in size, and so on. This process continues to the narrowest part of the gap between the roller and the grinding ring. The continuous and progressive size reduction of the material is accompanied by an increase in its specific surface.
- **Compaction of the bed of material:** In conjunction with the reduction in size there occurs intensive spatial rearrangement of the individual particles under crushing load. The compressive and shearing forces associated with this have a further size reducing effect, mainly by attrition, which is indeed the key factor in achieving fine pulverization in a roller mill. It is assisted by a certain amount of relative movement – depending on mill design features – between the rollers and the grinding ring. This relative movement also helps to prevent build-up on the ring if the mill is fed with moist or sticky material.
- **Depth and condition of the bed of material:** As explained, final size reduction in a roller mill is achieved substantially by attrition, i. e., the rubbing together of the material particles subjected to compression and shear while

undergoing rearrangement of their positions in the bed. To accomplish this requires the fulfilment of several conditions:

- sufficiently high specific grinding pressures;
- sufficiently large number of points and areas of contact of the particles with one another;
- sufficient possibility of movement of the particles in relation to one another.

These conditions are directly interrelated. If the bed of material increases in depth, the specific pressure exerted on the material, for a given pressure applied by the rollers, becomes less. If the depth of the bed decreases, the specific pressure increases, but the scope for relative movement of the particles is restricted and the number of their points and areas of contact is reduced. Hence every bed of material in a roller mill must be a compromise between the specific grinding pressure that pulverizes the material and the bed depth needed for achieving the product fineness required.

In most cases, if the mill is fed with material which is uniform in its granulometric composition and size reduction properties and which develops sufficient friction, a stable bed of more or less constant depth is formed on the grinding ring. With difficult materials there is scope for modifying and controlling the depth of the bed by dam rings or other such devices. If the feed material is too dry and has a high content of fine particles, stabilization of the bed may be achieved by moistening it.

It has been found that for the grinding of relatively soft materials, such as marl, the addition of high-grade hard limestone – required primarily for correction of the deficient chemical composition of the raw material – improves the performance of

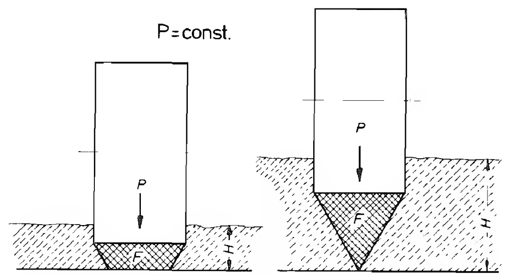

Fig. 43: Effective area of material subjected to pulverizing action during a roller pass, depending on bed depth

roller mills in terms both of throughput and of operational behaviour. To achieve such improvement, however, the limestone should be as coarse as possible within the maximum feed size limit that the mill can accept. The beneficial effect is due to the fact that, in the bed consisting largely of softer and finer particles including a very high proportion of recycled classifier rejects that have already been comminuted, the coarse limestone particles act as individual "hard spots" that offer higher resistance to the rollers and cause them to lift slightly. The rollers with their mechanical or hydropneumatic spring action then fall back onto the bed and do correspondingly more size reduction work on the finer particles they then encounter. Moreover, these hard spots promote more intensive spatial rearrangement of the particles of material in the bed and thus help to loosen it up, which likewise makes for more effective fine pulverization.

In general it can be stated that with feed material which may cause difficulties on account of low friction due to its specific material properties and/or granulometric composition it is possible to achieve distinct improvements in mill throughput, operational behaviour and specific power consumption by the addition of hard coarse particles. Improvements can similarly be obtained when dealing with feed material that tends to become solidly compacted on the grinding ring because of its moisture content and composition, e. g., too high a proportion of clay

Grinding speed; time of roller passage

In addition to the factors so far discussed – specific friction of the feed material, ratio of roller diameter to feed size, depth of the material bed, specific grinding pressure applied, composition of the material – the order of magnitude of the grinding speed is another important factor that governs the size reduction process in a roller mill.

The grinding speed is determined by the dimensions of the grinding ring and the magnitude of the centrifugal force needed for transporting the material. Apart from minor differences bound up with individual design features of the various mills, the grinding speed is much the same in all the usual roller mills for any given grinding ring diameter. To increase the grinding speed by some substantial proportion is of little benefit, because the larger centrifugal force that is then developed will shorten the residence time of the material on the roller path. Besides, because the time of roller passage – i. e., the time during which any particular particle of material is subjected to the action of the roller – is reduced, the available grinding pressure cannot be so effectively utilized for breaking down the particles.

It is known from materials testing technology that when compressive loads are applied at substantially higher speeds (rates of stress increase) than those employed in normal strength testing, distinctly higher crushing strengths are measured. In roller mills operating with the usual grinding speeds and pressures the rates of stress increase to which the material particles are subjected are very many times greater than those in compressive strength tests. Further increases in grinding speed would only increase the comminution resistance of the material even more and thus serve no useful purpose.

Börner has given a characteristic value k which expresses the time of action of the

grinding pressure (contact force per effective unit area) and provides a criterion for comparing roller mills differing in design:

$$k = \frac{z \times P}{v \times a} \text{ [kg} \times \text{second/m}^2\text{]},$$

where.

z = number of rollers [–]
P = total contact force [kg]
v = angular velocity × rolling circle radius [m/second]
a = effective width of rollers [m].

The effective width of conically tapered rollers can be taken as 100% of the actual width of the contact surface, while for rollers with convex surfaces about 60% may be adopted. For the latter, a more precise value can be found by examining the extent of wear on the rolling surface.

4.2.3 Control of roller mills

On account of the short residence times of the feed material in a roller mill – for example, a cycle time of about 30 seconds was measured in one such mill – these mills respond much more rapidly than tube mills to disturbing influences, e.g., variations in feed rate, grindability or moisture content of the material to be ground.

During the short cycle time in the mill the material is either on the grinding bed or is in suspension in the stream of air. Any influences that affect the residence time of the material on the bed will therefore quickly also manifest themselves in the change in dust concentration of the conveying air that sweeps through the mill. As the entire recirculation of material in nearly all these mills is effected entirely by pneumatic conveying action, it is directly associated with a pressure drop of the air. The pressure drop within the mill therefore, on the assumption of a constant volumetric rate of flow, constitutes an important controlled variable. By varying the feed rate and/or the pressure exerted by the rollers it is possible to keep the pressure drop at a constant value and thus to achieve a fairly uniform rate of classifier loading.

Besides the pressure drop, in combined grinding and drying mills the temperature in the grinding chamber and the rate of exhaust gas discharge are used as controlled variables.

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5 Grinding and drying of coal

5.1 Preparation of the coal, general considerations

With the steep rise in cost that fuel oil and natural gas have undergone since the early 1970s there has been a return to coal for industrial firing systems, including the kilns of the cement industry. This trend is reflected in the extensive literature that has appeared on the subject of pulverized fuel (coal and lignite), dealing with process engineering and also very extensively with safety engineering experience and requirements associated with the operation of coal grinding and drying plants.

The preparation of coal in the cement works – as distinct from its preparation in central plants which supply pulverized fuel ready for firing to industrial consumers and which do not come within the present scope – comprises the grinding and drying of the raw coal delivered to the works. In cases where coal consumption rates are high and coal from different sources of supply is used, it may be advantageous to blend the various coals in conjunction with stockpiling, so as to obtain a resulting fuel that is physically and chemically as closely uniform as possible and thus to achieve well balanced kiln operating conditions.

As a rule, for reasons of environmental protection and safety, cement works operate with relatively small coal stocks if they can rely on regular deliveries. Under these circumstances no elaborate storage installations are required. Stocks corresponding to about 30 to 60 days' consumption are normally held at the works.

Information on pulverized fuel firing systems is given in Section D.III "Firing technology".

5.2 Storage

Coal has the property of absorbing oxygen from the air. This is associated with heat evolution. If the heat cannot be given off at a sufficiently rapid rate to the surroundings, self-ignition may occur over a prolonged period of storage during which the temperature gradually rises to above the critical value of about 70°–80° C. The danger of self-ignition is especially great in coal that has come fresh from the mine and also in coal that has been crushed, so that a substantial increase in reactive surface area has occurred.
The self-ignition tendency is greater according as the volatile content of the coal is higher and also, because of the larger reactive specific surface, as its percentage of fine particles is higher.
Special safety precautions are not necessary for coal that is to be stored for only a few days, as in transfer or transhipment stockpiles. For longer-term storage, however, the coal should be deposited in layers which are each well compacted with the aid of rollers or crawler-mounted vehicles, so as to minimize the entry of atmospheric oxygen to the interior of the pile. Alternatively, the coal should be deposited in a thin layer and as loosely as possible, so that the heat evolved by oxidation can be quickly dissipated [22].

5.3 Grinding and drying

The raw coal, which generally has a moisture content of between 4 and 12% by weight in the as-supplied condition, is normally dried in combination with grinding in the mill. If coal slurry with a water content in the range from 15 to 30% is used, however, separate preliminary drying in a rotary dryer will be necessary before grinding and final drying in the mill are possible.
As a rule, in conjunction with grinding, the coal is dried to a residual moisture content of between 0.5 and 1.5%, which is suitable for firing. Completely dry coal is more difficult to ignite. In systems with intermediate storage of the pulverized coal it is, however, preferable to reduce the moisture content to below 1% in order avoid possible trouble with build-up (caking) and difficulties at bin discharge outlets, rotary gates and screw conveyors.
The fineness to which the coal should be ground for firing will depend on its flammability and its combustion rate. These properties are in turn governed by the content of ash and volatile constituents. Coal with a low volatile content will in general have to be ground finer than coal with a high volatile content. Commonly applied fineness criteria are: 10–15% by weight retained on the 0.09 mm and 1–2% by weight on the 0.2 mm standard sieve (DIN 4188 sieves). As an approximate guiding value the required fineness of the pulverized coal is expressed by the following rule of thumb. the percentage by weight retained on the 0.09 mm sieve should be equal to between 0.5 and 0.7 times the percentage volatile content (dry, ash-free) [24]. This will ensure good combustion with a short flame.
According to this rule, coal with a 30% volatile content would have to be pulverized to a fineness of 15–21% on 0.09 mm. In practice, however, it is preferable not to exceed 15% retained, even if the volatile content is fairly high, as this greater fineness of the coal is desirable to ensure complete combustion. This is

especially relevant to high-ash coal. On the other hand, for firing in a (pre)calciner associated with the preheater system it is quite appropriate to use a more coarsely pulverized coal, as experience has shown [24].

5.4 Grinding process

With regard to the functional coupling of the coal grinding plant with the firing operation, various grinding/drying systems have been developed which are not always very consistently designated by the terminology used in the technical literature.
In principle, a distinction can be drawn between the direct firing system and the indirect system. In the former, the pulverized coal is fed direct from the grinding mill (with reference to fuel grinding it is often called a "pulverizer") to the burner, the coal being carried in a stream of air which passes through the mill and is supplied as primary air to the kiln. On the other hand, in the indirect system the pulverized coal, separated from its carrying medium, is temporarily accommodated in an intermediate storage bin, from which it is fed independently to the burner.
The **direct system** in its basic form is shown schematically in Fig. 44. The pulverized coal is, as already stated, fed direct to the kiln, without intermediate storage. The hot air or gas needed for drying the coal in the mill may be available as exit gas from the kiln or exhaust air from the clinker cooler; alternatively, it may be supplied by a hot air generator (air heater). The mill system fan draws the hot air or gas (which may have a temperature not exceeding 350° C) through the grinding mill and discharges it, together with the pulverized coal it carries, as primary air to the kiln burner. This fan therefore functions also as the primary air fan The diagram shows that with this system the entire gas flow – comprising the hot air or gas, the water vapour driven out of the coal, and the "false" air that inevitably infiltrates into the plant – is thus supplied to the kiln.
The advantages of the direct firing technique are its simplicity in terms of layout and equipment, with correspondingly low capital expenditure, and its operational reliability, because there is no pulverized coal to be stored, nor any dust-laden exhaust gas to be dedusted. A disadvantage, however, is the high rate of primary air

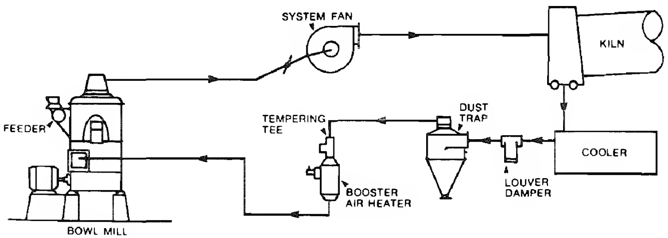


Fig. 44. Direct firing system (C. E. Raymond)

flow, resulting in correspondingly higher heat consumption of the kiln. Also, from the process engineering standpoint, the direct coupling of the mill to the firing system is unfavourable. The throughput of the mill has to be varied to suit the requirements of the kiln at any given time, so that optimum settings for the mill are generally not possible. Another drawback is that the operation of the kiln is dependent on that of the mill. Malfunction of the mill results in shutdown of the kiln, as does any interruption in the supply of raw coal to the mill, since there is no stored quantity of pulverized coal to serve as a buffer supply to bridge over any temporary breaks in the continuity of fuel output from the mill. Any variation or irregularity in the functioning of the mill will directly affect the firing system and thus the operation of the kiln.
A more sophisticated version of the direct firing principle is schematically illustrated in Fig. 45. Here the coal is ground in an air-swept ball mill. The pulverized coal is, however, collected in a cyclone; the mill system fan handles the substantially dedusted exhaust air, and this cleaned gas is supplied to the primary air fan. Part of the mill exhaust air remains as circulating air in the grinding system. This variant is a little more elaborate and expensive than the preceding one: the cyclone separator, which supplies the pulverized coal to the burner, has a damping effect on the transmission of any variations in performance or output from the mill to the burner. This technique is the **semi-direct firing system.** It is a somewhat comprehensive designation which includes a number of variants.
For instance, the semi-direct system shown in Fig. 46 is suitable for the grinding and drying of coal with a high moisture content [14]. It is more particularly advantageous when the quantity of hot gas that has to be passed through the mill in order to drive out the moisture is greater than the quantity of primary air that the kiln burner can accept. The surplus exhaust gas from the mill is discharged into the

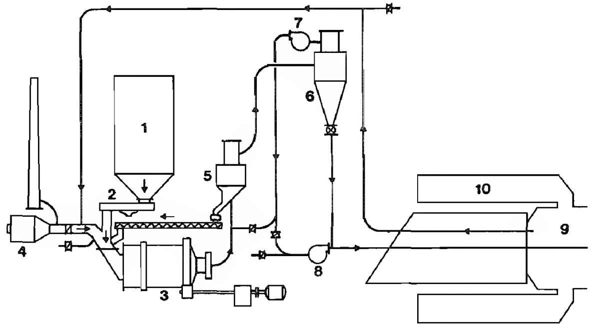


Fig. 45: Direct firing system (F. L. Smidth Tirax Mill)
1 bin for raw coal, 2 weight belt feeder, 3 air-swept mill (Tirax), 4 air heater, 5 air separator, 6 cyclone, 7 air circulating fan, 8 primary air fan, 9 rotary kiln, 10 planetary cooler (Unax)

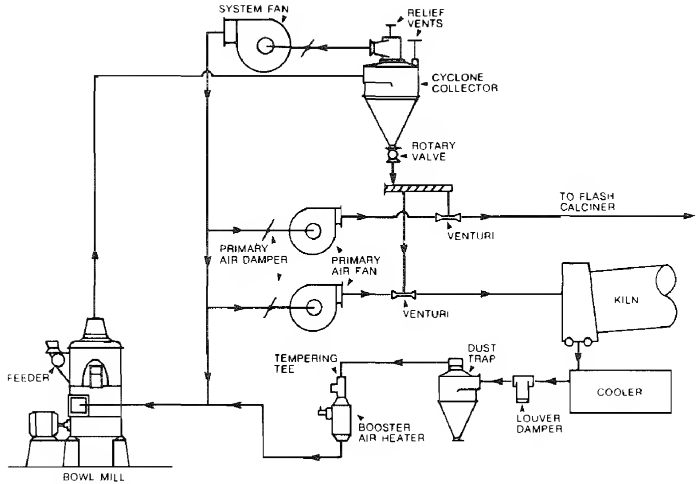


Fig. 46: Semi-direct firing system (C. E. Raymond)

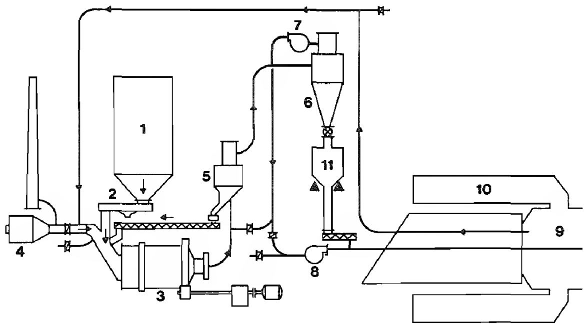


Fig. 47: Semi-direct firing system (F. L. Smidth)
1 bin for raw coal, 2 weigh belf feeder, 3 air-swept mill (Tirax), 4 air heater, 5 air separator, 6 cyclone, 7 air circulating fan, 8 primary air fan, 9 rotary kiln, 10 planetary cooler (Unax), 11 surge bin

atmosphere through a dust collecting filter. The filter, of course, constitutes an extra expense and is moreover a potential source of fire or explosion hazard.
The air-swept ball mill coal grinding plant shown in Fig. 47 is also an example of semi-direct operation. A surge bin of limited storage capacity is mounted on load cells which serve to control the rate of coal feed to the ball mill. In this arrangement, too, air is recirculated to the mill, and a quantity of exhaust air equivalent to the hot air supplied to the system is used as primary air.
In the grinding plant shown in Fig. 48, which serves two burners, there still exists the operational coupling of kiln and coal grinding mill. This semi-indirect plant supplies fuel to the kiln burner and to the precalcining burner in the preheater. Both burners are supplied with pulverized coal from a bin of limited storage capacity.
Fig. 49 shows yet another semi-direct firing variant. The raw coal is fed to the roller mill through a controllable feeder. The pulverized coal is collected in a cyclone and delivered through a rotary gate to a bin which is mounted on load cells and controls the set point of the mill feeder. The mill system fan is installed after the cyclon. The exhaust air from the cyclone, still containing a certain amount of fine dust, is used partly as primary air for combustion and is partly returned, mixed with fresh hot air, to the air-swept mill.
In general, if direct firing is used for two or more consumer units – more particularly cement kilns – it will be necessary to use two or more coal grinding mills to achieve suitably trouble-free plant operation.
The **indirect system** is characterized by the interposition of a substantial storage capacity between the coal grinding mill and the consumer equipment, which may comprise one or more burners. These are decoupled from the mill. Thus, one

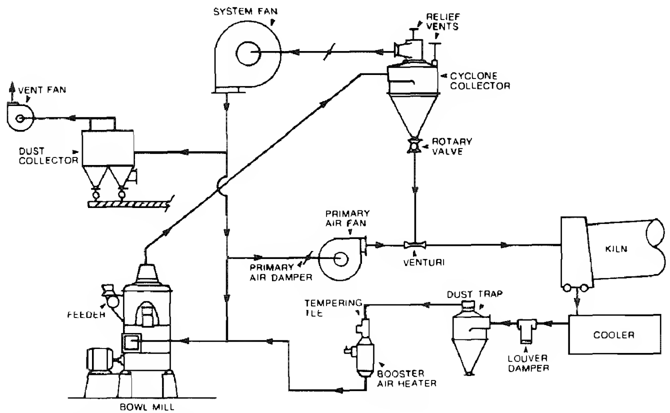


Fig. 48: Semi-direct firing system (C. E. Raymond)

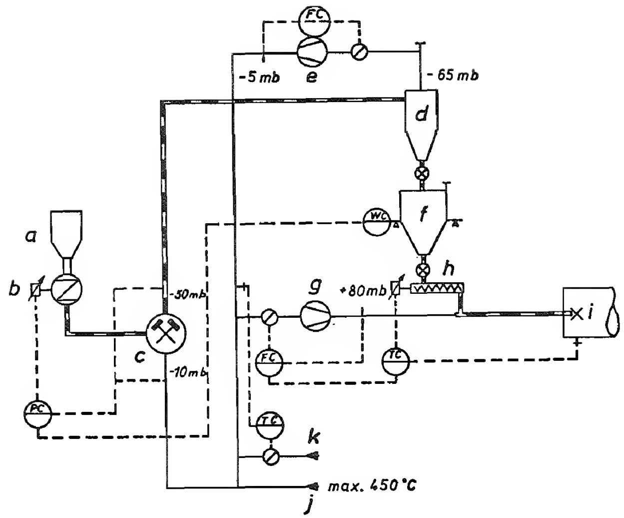


Fig. 49: Semi-direct firing system (Loesche GmbH)
a bin for raw coal, b coal feeder, c coal grinding mill, d cyclone, e mill fan, f pulverized coal bin, g primary air fan, h pulverized coal feeder, i kiln burner, j hot air, k cold air

centrally installed mill of appropriate throughput can supply the fuel requirements of several kilns. Such a pulverized fuel system is therefore sometimes referred to as a central grinding plant.
Exhaust air from the clinker cooler or preferably (because of its inert character thanks to its low oxygen content) exit gas from the kiln is used for drying the coal in the mill.
A central grinding plant equipped with a fabric filter is shown schematically in Fig. 50. The exhaust gas, with a temperature of about 80° C, is discharged into the external atmosphere. Fig. 51 shows a central grinding plant in which an air-swept ball mill is the pulverizing unit. The hot gas for coal drying is taken from the firing hood of the kiln. The mill exhaust gas is drawn through the system fan installed after the cyclone and is then divided into two flows, one of which is recirculated to the mill, while the other is discharged into the dust collector and thus to the atmosphere.
A solution in which the exhaust air from the mill is supplied as cooling air to the clinker cooler and which therefore does not require a dust collecting filter is shown in Fig. 52.

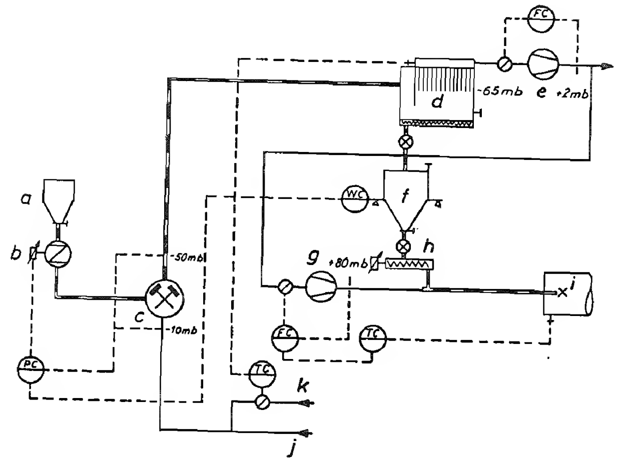


Fig. 50: Central grinding plant (Loesche GmbH)
a bin for raw coal, b coal feeder, c coal grinding mill, d fabric filter, e mill fan, f pulverized coal bin, g primary air fan, h pulverized coal feeder, i kiln burner, j hot air, k cold air

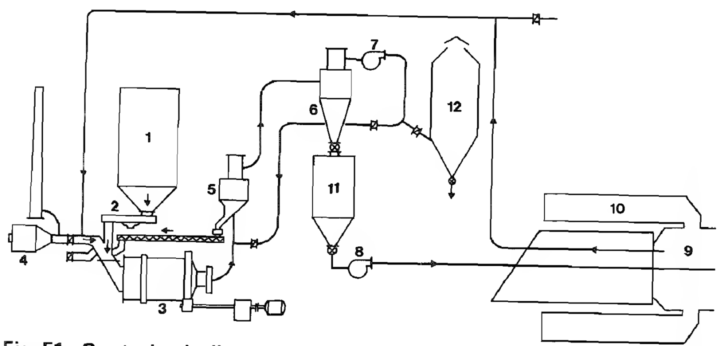


Fig. 51: Central grinding plant (F. L. Smidth)
1 bin for raw coal, 2 weigh belt feeder, 3 air-swept mill (Tirax), 4 air heater, 5 air separator, 6 cyclone, 7 air circulating fan, 8 primary air fan, 9 rotary kiln, 10 planetary cooler (Unax), 11 pulverized coal bin, 12 dust collector

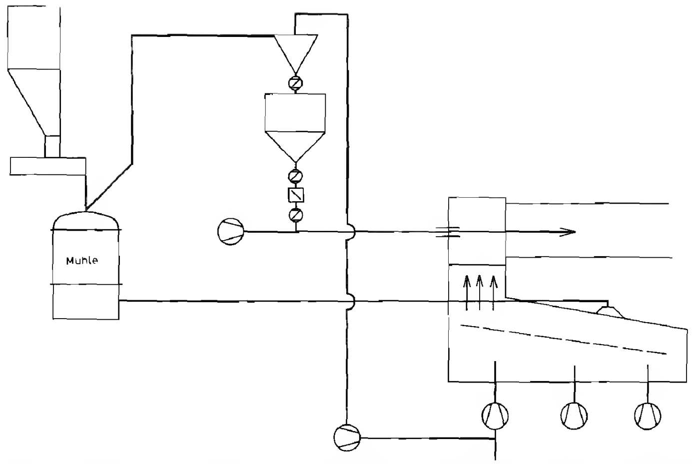


Fig. 52: Central grinding plant with exhaust air discharge into clinker cooler, requiring no dust filter (Heidelberger Zement AG, employee's invention, patents already granted in individual countries, applied for in others)

Advantages of the indirect system are: operational independence of coal grinding and kiln firing with regard to each other, possibility of supplying several consumer units from one central grinding plant; possibility of choosing the optimum rate of supply of primary air to the kiln; greater accuracy of feeding the pulverized coal to the burner, with shorter control dead time.
There are some disadvantages, however: higher capital cost of the equipment, which is more elaborate in terms of mechanical installations, control technology and safety arrangements; the need for a filter with a high dust collecting efficiency; the need for creating inert conditions as a safety precaution.

5.5 Types of coal grinding mill

The mills used for coal grinding and drying are either tumbling mills or roller mills. Some commonly employed types of mill will now be briefly described, without laying claim to completeness.

5.5.1 Tumbling mills

The tube mill or ball mill is especially suitable for the indirect firing system, i.e., where there is no direct connection between mill and kiln and where the pulverized and dried coal is stored in an intermediate bin of ample capacity. Thus the mill can be operated economically at a constant optimum rate of throughput, independently of the demands of the burners fed by it.

The ball mill is insensitive to foreign bodies in the feed material, and the wear of the grinding media can be compensated without any great effort or cost. The relatively long residence time of the coal in the mill has the effect of equalizing any short-term variations in the quality of the mill feed, thanks to the blending action of the system. Also, harder constituents such as quartz and pyrite are effectively pulverized
Ball mills for coal grinding are almost invariably operated as air-swept mills. As a rule, in order to cope with the relatively high moisture content of the raw coal, the mill is preceded by a drying compartment. The mill is mounted in trunnion bearings, usually at both ends. An advantageous alternative system of mounting that enables larger quantities of gas to be introduced into the mill is the sliding shoe bearing (Fig. 53).

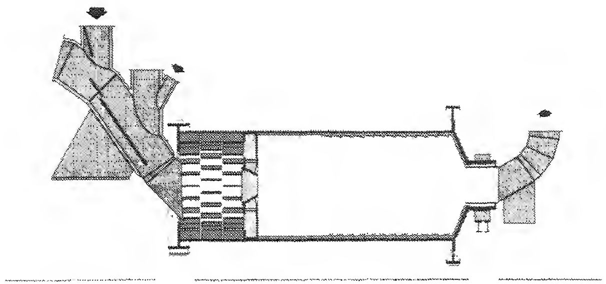


Fig. 53: Air-swept tube mill with drying compartment and sliding shoe bearing at inlet end (Krupp-Polysius)

5.5.2 Roller mills

As already noted in Section 4, the designation "roller mill" is often used as a generic one, comprising mills in which the grinding elements may not only be various types of roller, but may alternatively consist of balls. An advantageous feature for direct firing systems is the short residence time of the material in these mills, so that mill operation can be quickly adjusted to suit the firing requirements at any given time. Economically advantageous is moreover the fact that the power consumption of a roller mill drive is more closely dependent on the rate of material throughput than that of a tumbling mill. The throughput control ratio is about 1 2 in all types of roller mill.
Quartz and pyrite are frequently present in coal. They cause a higher rate of wear of the grinding elements, so that more frequent renewal of these parts is necessary

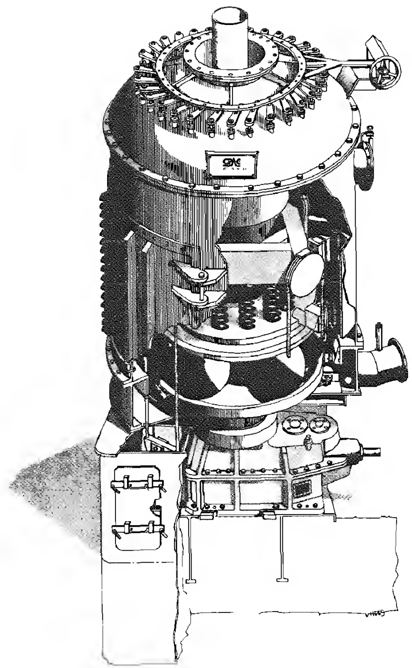


Fig. 54: Ring-ball mill for coal grinding; standard type, designed to resist pressure surge (Claudius Peters)

and the operational availability of the plant is correspondingly diminished. This is obviously a drawback in direct firing with close interconnection of mill and kiln.
The Claudius Peters direct-firing mill is a ring-ball mill which is available in two versions for operation under inert internal atmosphere and designed to an explosion-resistant specification so that it can withstand pressure surges of 3.5 bar or 50 psi (Fig. 54).
The Krupp-Polysius RMK roller mill can be supplied with a housing designed to resist pressure surges of up to 8 bar. This range of coal grinding mills comprises throughputs from 2.3 to 62 t/hour for a Hardgrove grindability index of 55 and a product fineness corresponding to 12% retained on the 0.09 mm sieve (Fig. 55)
The Atox coal grinding mill is a fairly new development of the firm of F. L. Smidth (Fig. 56). It has a flat-topped grinding table, and the three grinding rollers are each mounted on a shaft which is attached to a central yoke. The mill is designed to

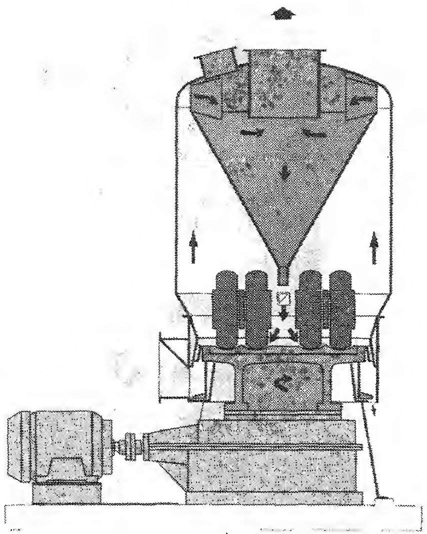


Fig. 55: Roller mill for coal grinding (Krupp-Polysius)

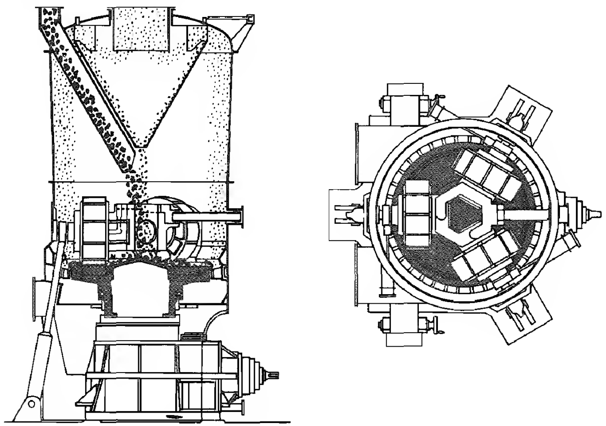
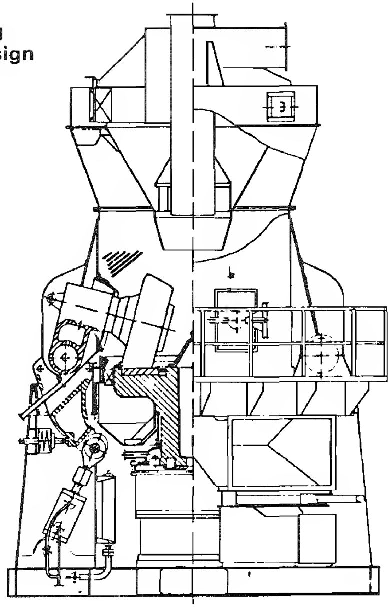


Fig. 56: Atox roller mill for coal grinding (F. L. Smidth)

Fig. 57: Three-roller direct firing mill LM 26.30 D, of modular design (Loesche GmbH)



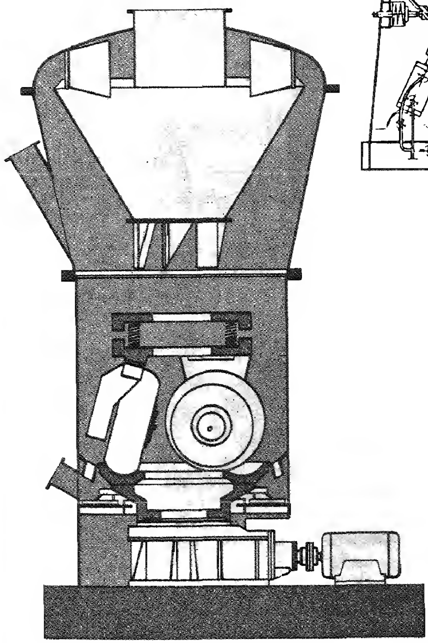
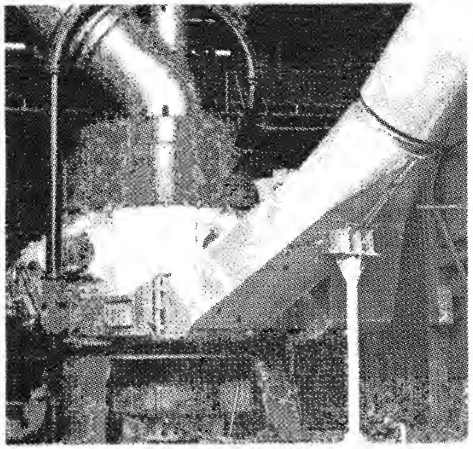


Fig. 58: MPS roller mill (Gebr. Pfeiffer)



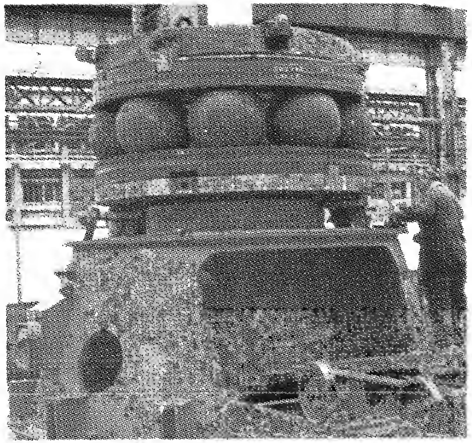


Fig. 59: Type "E" ring-ball mill (Fives-Cail Babcock)

comply with the United States and European safety codes for resistance to explosion pressure surges. The mills in this range have throughputs from 5.5 t/hour (drive motor power rating 55 kW) to about 80 t/hour (800 kW), their product having a fineness corresponding to 10% retained on the 0.09 mm sieve.

The roller mill originally developed by the firm of Loesche for coal grinding, and subsequently used also for the grinding of other materials, is at present available in two ranges intended more particularly for coal. The principal features of the range of smaller mills with their two grinding rollers and their grinding tables from 1300 to 1900 mm diameter are: throughputs from 14 to 40 t/hour, with corresponding drive power ratings from 112 to 330 kW, yielding a product ground to a fineness of 15% retained on 0.09 mm for coal with Hardgrove grindability index of 90. These mills are resistant to pressure surges of 3.5 bar, thus satisfying the conditions of the German VDI Code 3673. The larger coal grinding mills built by Loesche are characterized by modular design and have two, three or four rollers. This range starts with a mill designed for a throughput of about 40 t/hour (420 kW installed power) and equipped with a grinding table of 2100 mm diameter. See Fig. 57.

Other extensively used coal grinding mills are the MPS roller mill of Gebr Pfeiffer AG (Fig. 58) and the type E ring-ball mill of Fives-Cail Babcock (Fig. 59).

5.6 Safety requirements

Special requirements intended to ensure safe operation of coal grinding plants have to be fulfilled in order to eliminate explosion hazard. The potential existence of such hazard is due to the following factors:

- combustible materials in finely divided form are present;
- the dust (pulverized coal) concentration is within the explosive range, i.e., between the lower and the upper limit of flammability;
- oxygen is present in concentrations that can sustain explosions;
- sources of ignition may develop.

Even fairly coarse coal particles of about 1 mm size, suspended in air, can constitute an explosion hazard. In the grinding plant the pulverized coal is always present in ignitable fineness.

The explosive range for pulverized coal, or coal dust, suspended in air depends on its physical properties, such as its fineness and moisture content, and on its chemical composition, such as its ash content and volatile content. The lower limit above which the concentration of coal particles in atmospheric air is potentially hazardous thus varies according to circumstances. Values from 200 g/m^3 to as low as 15 g/m^3 in the most unfavourable case have been reported (Narjes 1963, Wibbelhoff 1981). Of course, the figures found by various investigators depend not only on the physical and chemical properties of the pulverized coal, but also on experimental conditions such as the ignition energy input.

The important fact, however, is that it is not economically possible to operate coal grinding systems with concentrations of pulverized coal which are consistently below the lower limit of flammability and thus "safe".

There is also an upper limit of flammability, which is located at concentrations of between 1500 and 6000 g/m^3, again depending on various circumstances. At

concentrations of coal suspended in air in excess of this limit there is considered to be no danger of explosion. During start-up and shutdown of a coal grinding plant the internal conditions always pass through the explosive range bounded by these two limits.
In terms of oxygen concentration the lower limit of flammability is around 14% by volume of the air in which the coal particles are suspended. A gas mixture containing less than this oxygen amount is regarded as inert with regard to coal dust explosion and therefore "safe".
Lowering the oxygen content in the grinding circuit has the effect of raising the lower limit of flammability and lowering the upper limit, so that the explosive range is narrowed. Also, with lower oxygen content the ignition temperature of the mixture of pulverized coal and air is raised, and this effect, too, tends to reduce the hazardous range of concentration.
The most dangerous source of ignition liable to initiate explosions are smouldering pockets that may develop in coal dust deposits inside the plant. Ignition of the dust may be brought about by too high a temperature of the gas used for drying the coal.
Hence the conditions for the occurrence of an explosion are at times fulfilled in a coal grinding plant. As it is not possible to eliminate deposits of combustible dust inside the plant, the required degree of safety is attainable only by using inert gas for drying and conveying the pulverized coal. In the event of failure of the supply of inert gas a potentially hazardous condition may still arise, so that, theoretically at least, it would be necessary to provide a separate and independent source of inert gas for immediate availability in an emergency.
In actual practice the grinding plant normally operated under inert internal conditions is designed to an explosion-resistant specification in that it is able to withstand pressure surges of a certain magnitude, while it is additionally provided with pressure relief venting, so that the consequences of an explosion are kept within acceptable limits and no serious damage is done. Venting devices are of various types: blow-out panels, explosion doors, etc. In the interests of safety, personnel should not be allowed to enter certain "no-go" zones near these devices while the plant is in operation. Maintenance, repairs and inspections of vital parts should be carried out only during plant shutdowns.
The principles and precautions applicable to coal grinding and drying are even more stringently applicable to lignite (brown coal), which is especially hazardous on account of its higher content of volatile matter.

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c) F. L. Smidth & Co. A/S, DK-2500 Valby Kopenhagen
d) Fuller Company, Bethlehem, Pennsylvania 18001
e) CE Raymond Combustion Engineering, Inc., Chicago, Illinois 60606
f) Loesche-GmbH, D-4000 Düsseldorf
g) O. & K. Orenstein & Koppel Aktiengesellschaft, Werk Ennigerloh, D-4722 Ennigerloh
h) Five-Cail-Babcock, Hauptverwaltung, D-4150 Krefeld
i) Gebr. Pfeiffer AG, D-6750 Kaiserslautern

II. Raw meal silos

By H. K. Klein-Albenhausen

1 General

For the manufacture of cement clinker it is necessary to prepare a raw mix fulfilling certain conditions as to its chemical composition (see Section CII2). Raw materials which already in their natural state conform to these requirements are exceedingly rare. In order to obtain a suitable mix, in modern cement production it is therefore standard practice to apply blending and homogenization of the raw materials at some point between the crushing plant and the raw mill This is normally done in a so-called blending bed – a stockpile which serves not only for storage of the crushed stone, but is so built up and equipped that preliminary homogenisation of its composition can be effected (see Section BII).
Homogenization also takes place during the grinding process. Although this further improves the chemical uniformity of the material, it is in most cases still not enough to meet the strict requirements of present-day cement burning (see Section C).
This being so, over the years various methods and systems have been developed which enable a high degree of raw meal homogenization to be achieved economically. Special silos equipped for storing and homogenizing the raw meal are available. The systems can be broadly subdivided into those with batchwise (intermittent) and those with continuous operation. Which system should be chosen in a given case will depend on circumstances and requirements. Also, besides chemical and technical considerations, the question of economy (cost of construction, operating expenses, etc.) must not be ignored.

2 Batchwise homogenization

With this system the raw meal in a large-capacity silo is completely fluidized by the admission of compressed air through suitable inlets in the bottom of the silo. The air penetrates the silo contents, thus greatly reducing or cancelling the friction between the particles (Fig. 1). An overall circulatory motion is obtained by

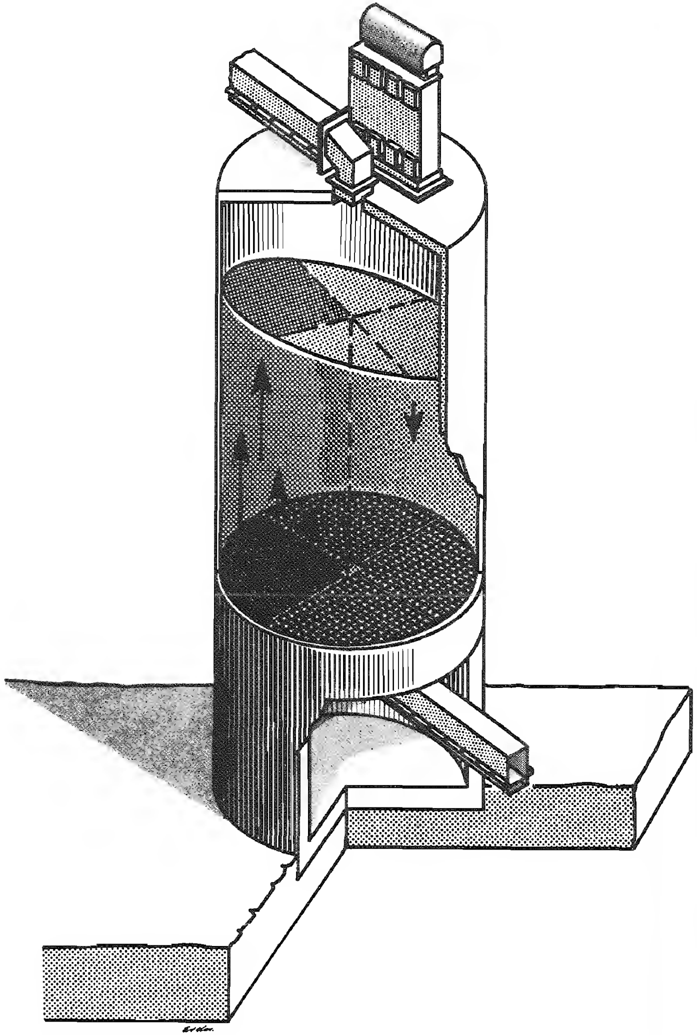

Fig. 1: Homogenizing silo embodying the quadrant system

admitting the air cyclically through different zones (e. g., sectors or segments) of the silo bottom. The greater part of the air enters the silo in the so-called active aerating zones, while in the other zones only so much air is supplied as to keep the material over them merely in a fluidized condition. With this method even very large and long-term variations in the chemical composition of raw meal can be reduced to very low amounts.

The actively aerated zones are switched systematically at regular intervals by means of special valve equipment, so that they move round and round the silo bottom – e. g., sector by sector – in a clockwise or anticlockwise direction. It is more particularly this continual progression of the active zones that keeps the contents of the silo in motion and effects the desired homogenization.

A homogenizing silo is generally designed to hold 10 to 12 hours' grinding output, so as to ensure sufficient treatment to cancel out the remaining variations in the chemical composition of the raw meal. The time required for achieving this will of course depend also on the degree of prehomogenization of the raw material ahead of the mill.

The height (depth) of material in the silo should not exceed 1.5 times the silo diameter. Normally a height/diameter ratio of 1.2:1 is adopted.

The specific air supply rate (m^3 of air per minute and per m^2 of aerated silo bottom area) will depend on the ease with which the material can be fluidized. For normal raw meal the required specific air rate is about 1 m^3/m^2 minute, with air supplied at a pressure of 2–3 bar.

These figures indicate that pneumatic homogenization demands a substantial energy input. It does, however, achieve a relatively high degree of homogenization, so that even quite large variations in the composition of the raw meal can be effectively reduced The result is a function of the homogenizing time and thus of the energy consumed. Fig. 2 shows an efficiency curve for a system of this type.

To compensate for the intermittent operation, two homogenizing installations may be employed, one being aerated while the other is supplying raw meal to the kiln. "Two-storey" construction – one silo mounted over the other – is commonly employed.

3 Continuous blending*)

As already mentioned, with modern quarrying methods and with the introduction of efficient blending beds a substantial degree of homogenization of the raw material is achieved already before it is supplied to the raw grinding mill. As a result, little or no homogenization of the raw meal may be necessary, in which case the raw meal silo will function merely as a buffer store. All the same, the raw meal composition will generally still show some residual variation, and it is advan-

*) In the literature no clear distinction is drawn between "blending" and "homogenizing", these terms often being treated as synonyms. Some authors, however, use "blending" where two or more recognizably different material components have to be merged or mixed or where more or less distinct layers of material are incorporated with one another

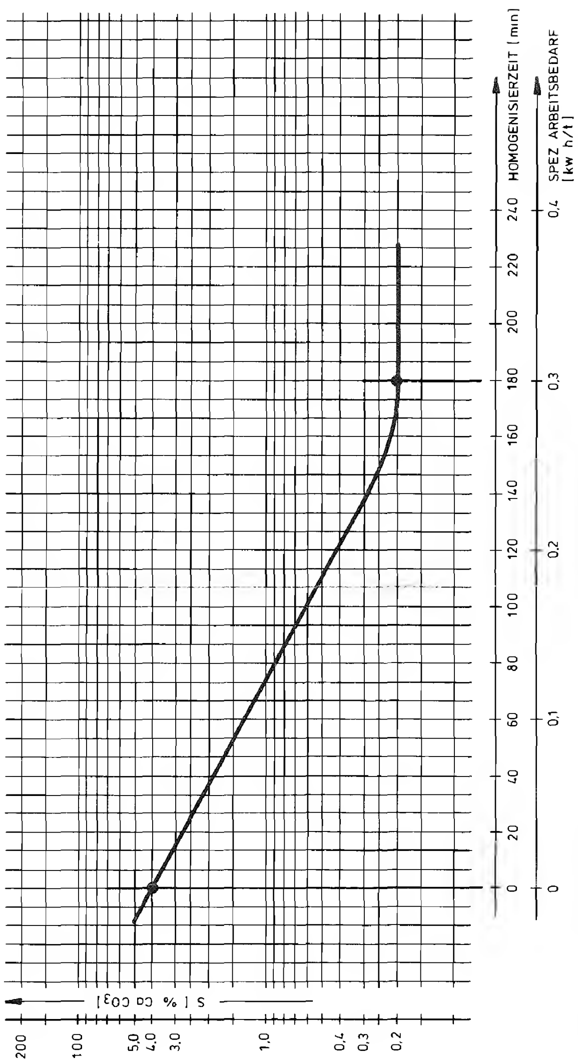

Fig. 2: Performance diagram of a pneumatic homogenizing silo
spez. Arbeitsbedarf = specific power consumption
Homogenisierzeit = homogenizing time

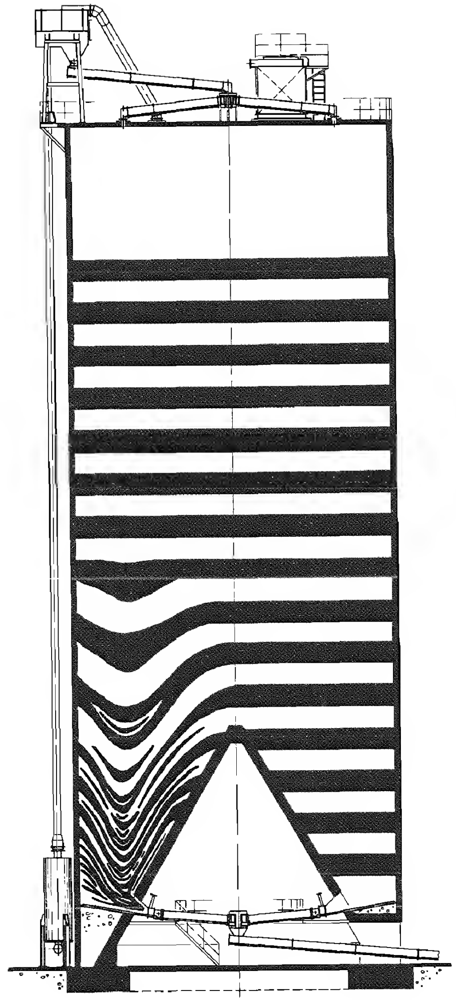

Fig. 3: Blending chamber silo (schematic)

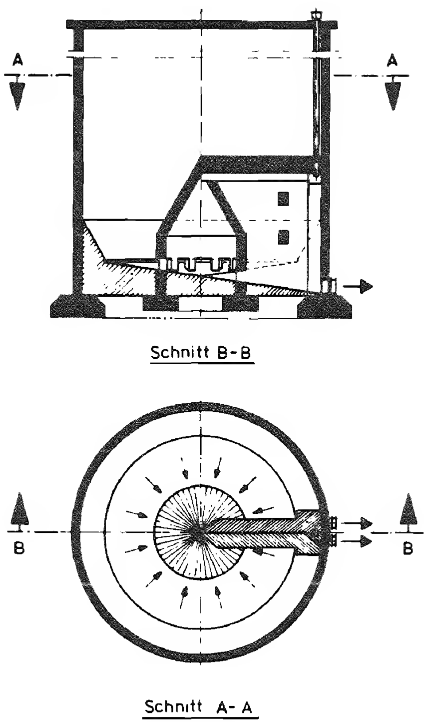

Fig. 4: Blending chamber silo (schematic)

tageous to reduce this as far as possible. In modern systems this is usually done in continuously operating silos equipped with special discharge aerating chambers, referred to as blending or mixing chambers, often conical in shape.
The raw meal is deposited layer by layer in the silo (Figs. 3, 4, 5). It enters through a system of troughs and several inlet openings so as to build up these layers as uniformly as possible over the entire cross-sectional area of the silo. The actual blending is effected during the emptying process, this being brought about by cyclic aeration of bottom sectors or zones in such a way that funnel flow develops, causing the respective layers to flow into the "funnel" cavity and merge. In order to prevent fresh raw meal from rushing prematurely into the funnel, this action must

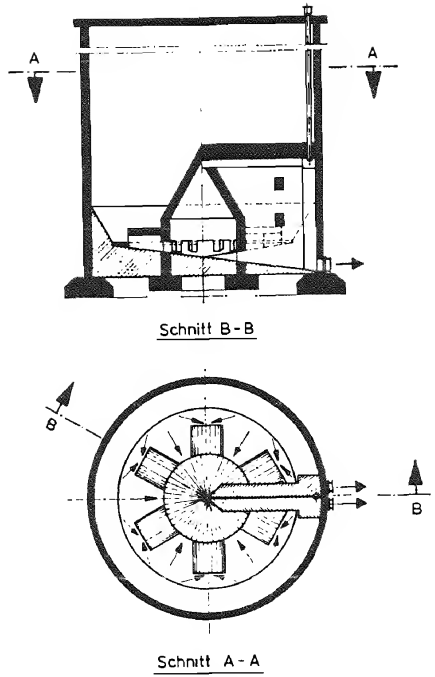

Fig. 5: Blending chamber silo (schematic)

be stopped from time to time and a new funnel be formed. This is done cyclically by an air distribution system or preferably by means of shut-off valves associated with the conical discharge chamber.
The blending efficiency of such an installation is inevitably limited and will depend substantially upon the manner in which the raw meal is deposited layerwise in the silo and how effectively the funnelling-down of the material to achieve the mingling of the layers is continually accomplished. It is reckoned that a homogenization factor of at least 3:1 is attainable with one such silo, and that 5:1 and higher can be attained if two silos are operated in combination with each other. However, the efficiency varies considerably from one silo system to another. Power consumption is relatively low – about 0.1 – 0.2 kWh per tonne of raw meal.

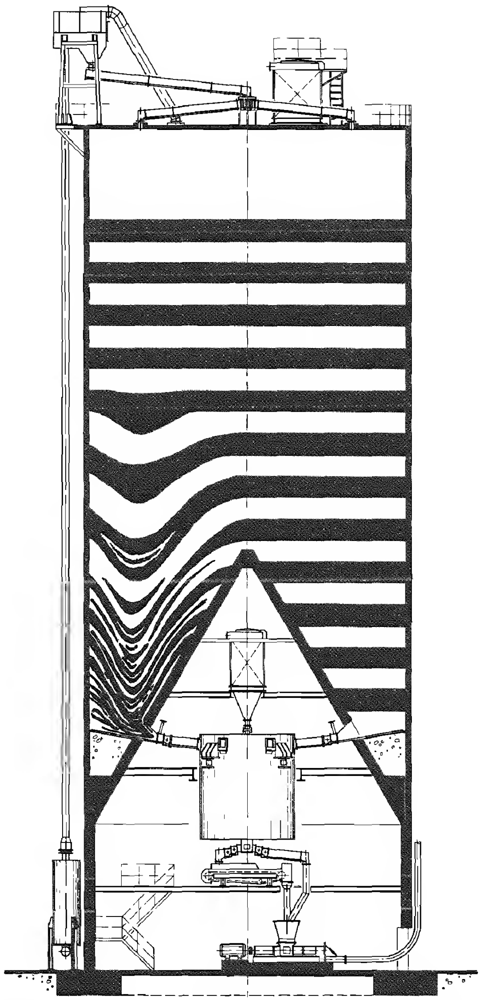

Fig. 6: Blending chamber silo with elevated homogenizing compartment (schematic)

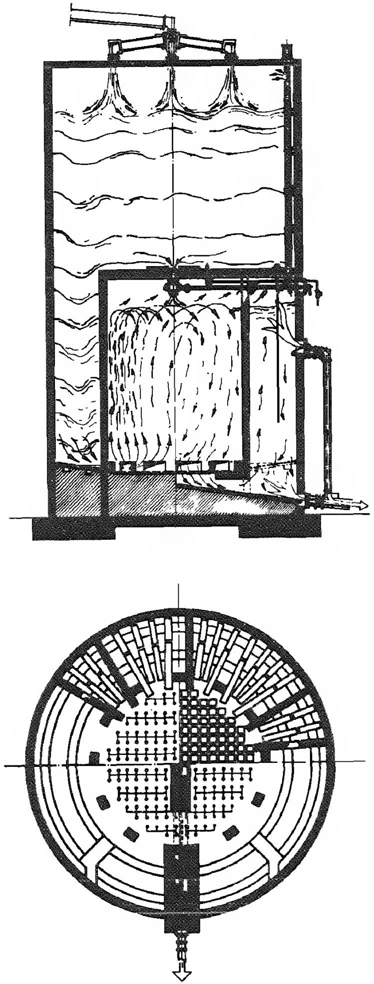

Fig. 7: Blending chamber silo with integral homogenizing compartment (schematic)

4 Combined systems

Raw meal silo installations embodying a combination of the two principles outlined above – batch homogenization and continuous blending – are so designed that the meal is prehomogenized (blending of layers) in a continuous silo and then passed to a comparatively small second silo for final homogenization.
As a result of the blending of the layers of material by funnel action in the continuous silo not only the maximum values but also the frequency of the variations are reduced, so that the final homogenizing treatment can be performed fairly quickly, which in turn means that the second silo need only have a correspondingly small volumetric capacity for attaining the specified uniformity in chemical composition of the raw meal (Figs. 6 and 7).
The small second silo is aerated continuously and is fed with material at a rate equal to the rate of discharge from this silo, which may take the form of a homogenizing chamber installed within the continuous blending silo and constituting an integral feature thereof.

5 Summary

Which type of blending/homogenizing silo system will provide the technically optimal and economically favourable solution for any particular cement works is a question that must be viewed in the overall context of the raw material conditions and preparation equipment envisaged. Various combinations of raw meal silos are shown in Fig. 8.

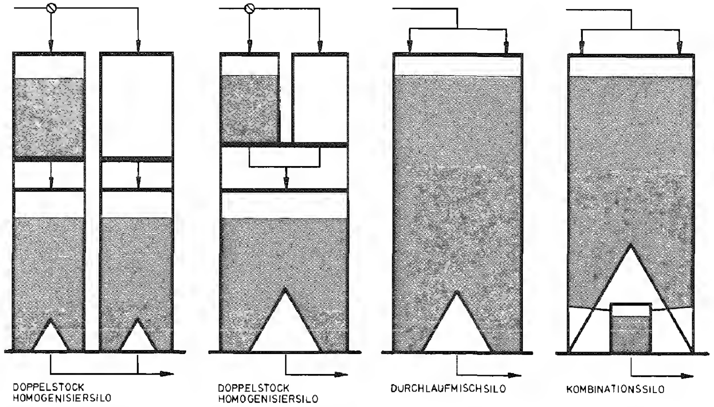

Fig. 8: Various raw meal blending silo installations: two-storey homogenizing silo, continuous blending silo, combination silo

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Acknowledgements for illustrations.

Fig. 1: IBAU HAMBURG
Fig. 2: IBAU HAMBURG
Fig. 3: IBAU HAMBURG
Fig. 4· CPAG
Fig. 5 CPAG
Fig. 6. IBAU HAMBURG
Fig. 7: CPAG
Fig. 8· IBAU HAMBURG

III. Cement burning technology

1 Kiln systems

By E. Steinbiss

1.1 Types of kiln

1.1 1 General

In the early days of cement manufacture the clinker was produced in shaft kilns (vertical kilns) which were manually charged and controlled It was a process involving strenuous physical labour and had the drawback of irregular operation, yielding a clinker of variable and often inferior quality. Besides, the capacity of such kilns was low.

This unsatisfactory system was superseded by the automatically operating shaft kiln. With good raw materials and suitable fuels it is thus possible to obtain regular kiln performance, but the disadvantage of limited output per kiln – not above about 300 tonnes per day – remains.

Late in the nineteenth century the rotary kiln was developed in Britain, introduced into the United States and, from that country, adopted in Continental Europe. With this kiln it had become possible to use any type of fuel: solid, liquid or gaseous (coal, oil, gas). The raw materials were introduced into the rotating tube in the form of "slurry" (wet process) or "raw meal" (dry process). In comparison with the shaft kiln, the capacity of the rotary kiln was soon greatly increased, especially after very effective homogenization methods, preheating and precalcining systems had been developed, and efficient measuring and control instrumentation had been introduced.

All these developments and improvements have helped to bring the rotary kiln to a high level of performance. Thus, clinker outputs of 3000 t/day are now regarded as perfectly ordinary, while kilns capable of producing 6000 to 8000 t/day are by no means very exceptional. Besides the development of large kiln units there has been a very notable reduction in specific heat consumption, which makes for greater

economy and is of course a desirable development in the general effort to conserve energy. All this has been achieved without detriment to the high quality standards with which the cement clinker has to comply.
In view of this evolution, the present chapter will be concerned only with rotary kiln systems.

1.1.2 Long rotary kiln

Feed: slurry with about 30 to 45% water content (wet process) or dry raw meal (dry process).
Shell diameter: up to about 7.0 m.
Length of kiln: for wet or dry process between 32 and 35 times shell diameter.
Inclination of kiln: 3.0 to 4.5%.
Rotational speed of kiln: 1.5 to 2.5 r.p.m., corresponding to a circumferential velocity of about 0.3 to 0.9 m/sec.
Refractory lining: see Section D.III.5.
Internal chain system: weight of chain fittings about 0.1 to 0.13 t/m^3 of effective kiln volume.
Thermal rating of refractory lining in burning zone: 20 to 25 $GJ/m^2 \cdot h$
Residence time of material in kiln: 3 to 5 hours.

1.1.3 Short rotary kiln

Feed: semi-dry or dry raw meal (semi-dry or dry process).
Shell diameter: up to about 7.0 m.
Length of kiln. between 15 and 17 times shell diameter.
Inclination of kiln: 3.0 to 4.5%.
Rotational speed of kiln: up to about 2.5 r.p.m.
Refractory lining: see Section D.III.5.
Thermal rating of refractory lining in burning zone: 20 to 25 $GJ/m^2 \cdot h$
Residence time of material in kiln: 40 to 60 minutes.

1.2 Method of support for rotary kilns

Depending on the length of the kiln, it is supported on two or more tyres (riding rings) mounted on carrying rollers. The ratio of roller diameter to tyre diameter ranges from 1:2.2 to 1:4.4 and depends on the kiln shell diameter and on the number of tyres and roller sets on which the kiln is supported, this in turn being a determining criterion for roller size with respect to permissible bearing contact pressure.

1.2.1 Rollers and their bearings

The centres of the carrying rollers are positioned at an angle of 30 degrees on each side of the vertical centre-line of the kiln shell cross-section. See Figs. 1 a, 1 b and 1 c. The spacing of the roller sets along the kiln will depend on the positioning of the tyres and on the longitudinal thermal expansion of the shell.

The thrust due to the slope of the shell and its rotation has to be resisted. On small kilns with rollers of correspondingly small diameter the latter are disposed with their axes slightly at an angle to the longitudinal axis of the kiln instead of parallel to it (Fig. 2). However, on present-day big kilns the axes of the rollers are placed parallel to the kiln, an arrangement which enables the kiln to perform continuous

Fig. 1 a: Kiln mounting
(from Labahn/Kaminsky, 1974)

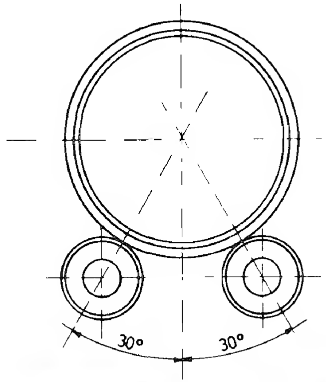
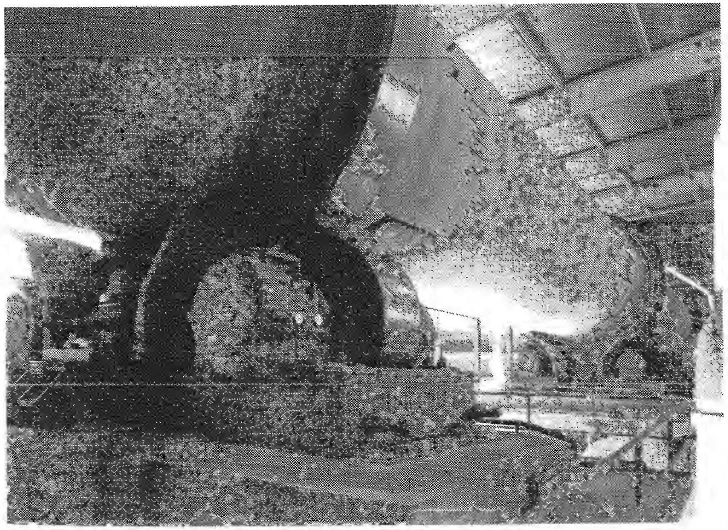


Fig. 1 b: Rotary kiln mounting
(KHD Humboldt Wedag AG, Cologne)



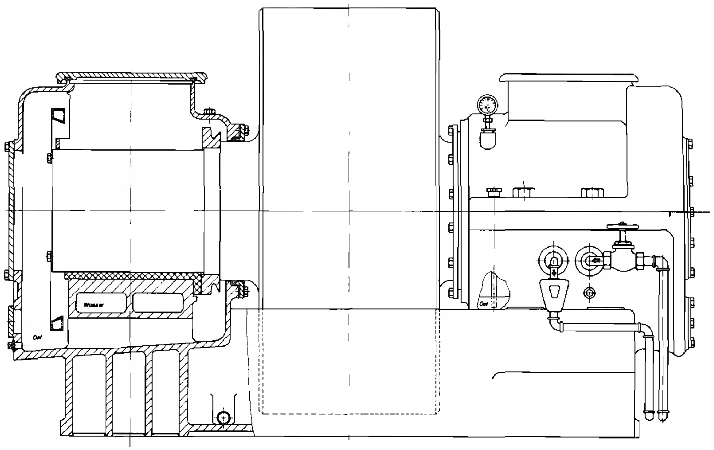
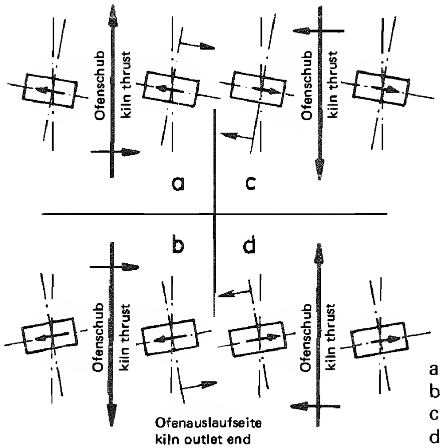


Fig. 1c: Bearing pedestal for kiln rollers (KHD Humboldt Wedag AG, Cologne)



a = kiln rotates clockwise
b = kiln rotates clockwise
c = kiln rotates anti-clockwise
d = kiln rotates anti-clockwise

Fig. 2: Kiln roller adjustment (from Labahn/Kaminsky, 1974)

"uphill" and "downhill" movements, while now the tyres will not bear obliquely on the rollers and thus not cause grooving and lateral deformation of them.
The carrying rollers are mounted in scoop-lubricated plain bearings, occasionally in roller bearings (Figs. 3 and 4).

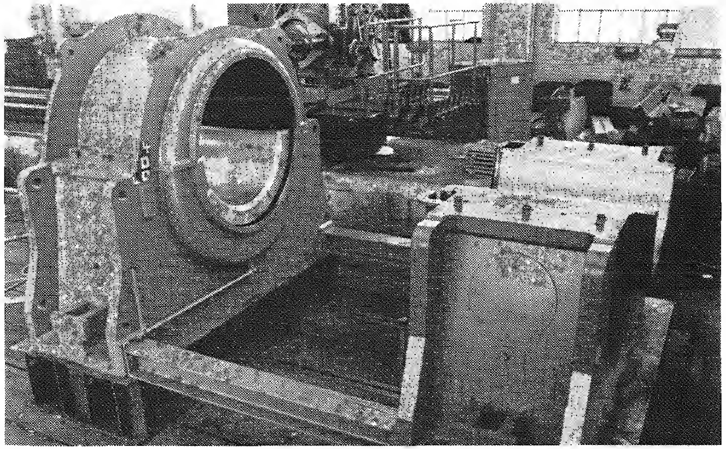


Fig. 3: Kiln roller mounting (KHD Humboldt Wedag AG, Cologne)

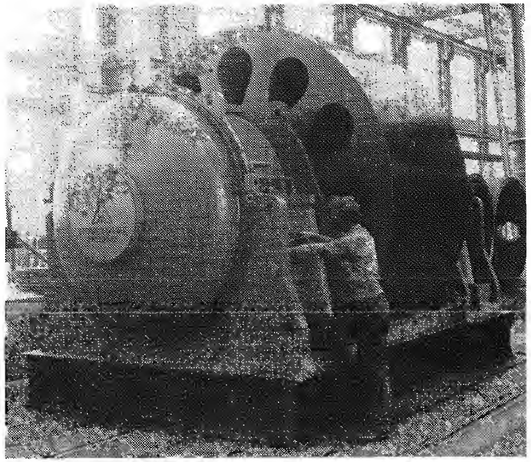


Fig. 4: Kiln roller with mounting (KHD Humboldt Wedag AG, Cologne)

1.2.2 Thrust rollers

In order to limit the uphill and downhill sliding movements of the tyres on the rollers, small kilns are provided with thrust rollers which are given a certain permanent setting to limit the range of movement. Large kilns are equipped with a more sophisticated system comprising hydraulically controlled thrust rollers (Figs. 5, 6 and 7).

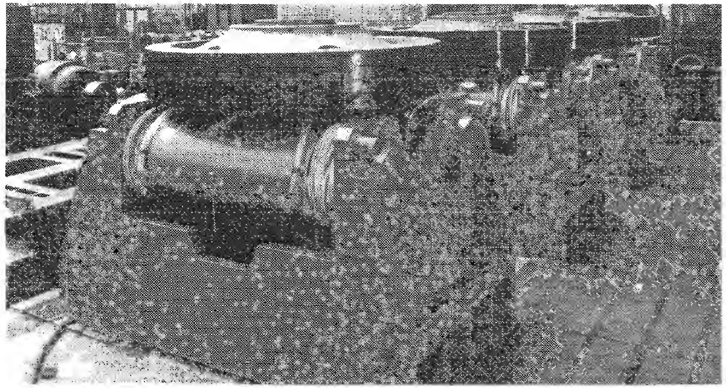


Fig. 5: Thrust roller (KHD Humboldt Wedag AG, Cologne)

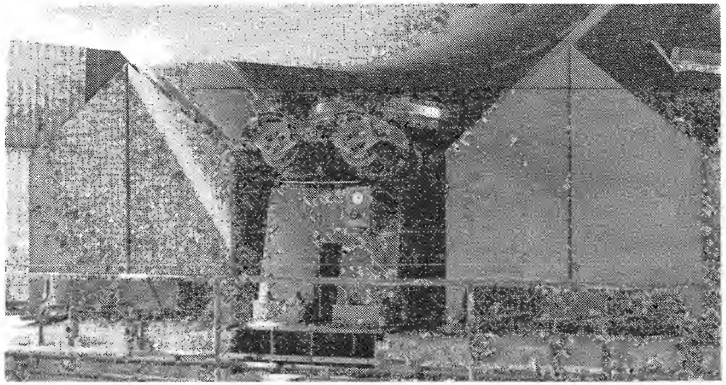


Fig. 6: Thrust roller set (KHD Humboldt Wedag AG, Cologne)

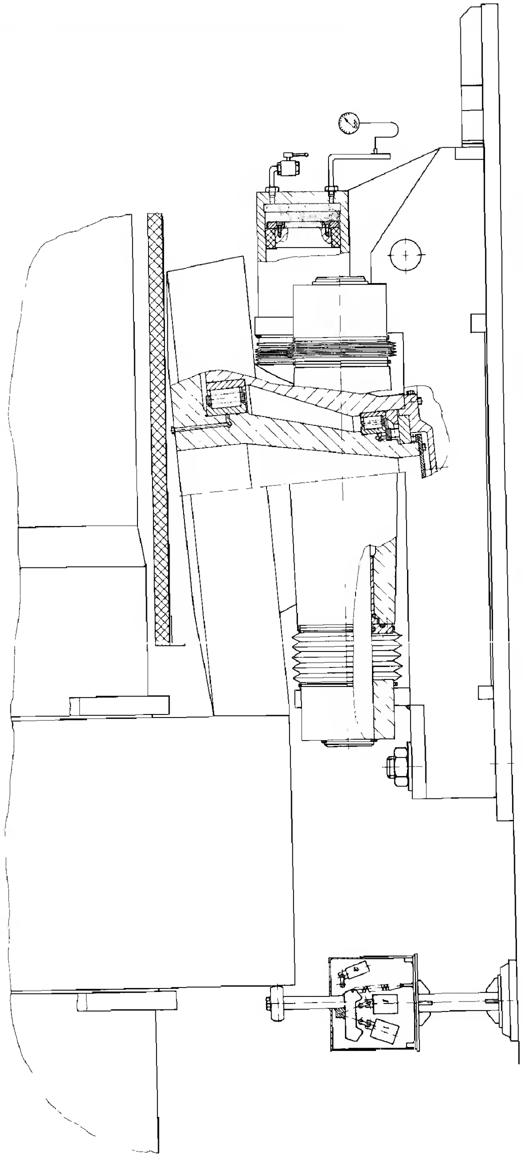


Fig. 7: Hydraulically controlled thrust roller system (KHD Humboldt Wedag AG, Cologne)

1.2.3 Tyres

The tyres (riding rings) are among the most important constructional features of a rotary kiln. They constitute the supporting elements which have to transmit the load of the kiln and its contents to the carrying rollers. This function has to be reliably performed despite longitudinal movements and thermal expansion of the kiln shell.

The internal diameter of the tyre must be sufficiently large to provide adequate clearance for the shell when the kiln has attained its full operating temperature. Insufficient clearance is liable to cause pinching and possible constriction of the shell by the tyre.

Generally speaking, the tyre should be so dimensioned in relation to the shell that the "ovalling" (elliptical distortion) of the latter remains less than 0.2% (as stated by Nies, 1942). The ovality can be measured on the rotating shell by means of the Shelltest apparatus. Damage to the refractory lining due to excessive cross-sectional distortion can be avoided by ensuring that the ovality measured in this way does not exceed the amounts indicated in Table 1.

Table 1: Permissible relative ovality values, as determined by the Shelltest method (according to Erni/Saxer/Schneider, 1979)

kiln diameter	m	3	4	5	6
ovality	%	0.3	0.4	0.5	0.6

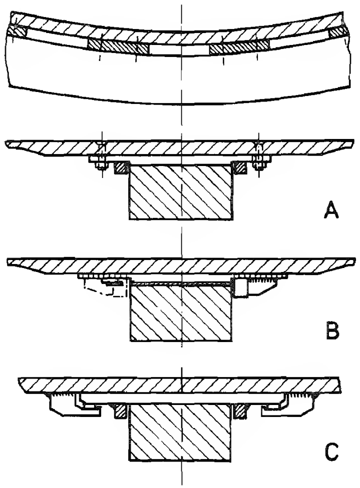


Fig. 8: Tyre mounting systems (from Erni, 1974)
A Bolted chairs
B Welded chairs, with wearing plate
C Guided chairs, keyed

Under normal operating conditions a clearance ranging from 3 to 20 mm is formed between the shell and the tyre, depending on the respective temperatures of these components. Because of this loose fit (so-called floating tyre) there occurs some circumferential slip or lag of the tyre in relation to the shell. In the axial direction (longitudinal direction of the kiln) the tyre is located in position by means of retaining elements welded to the shell. With floating tyres the shell ovality can be kept to acceptably low values only by sufficiently rigid shell and tyre construction, in conjunction with the least possible clearance compatible with avoiding the risk of the shell being constricted by the tyre. Since there remains an element of uncertainty, the tyre clearance or the circumferential lag of the tyre in relation to the shell should be continuously monitored. If the clearance is too large, it should be reduced by the insertion of filler plates (packings) between the shell and tyre. The require plate thickness can be calculated from: $p = u_{min}/\pi - 3$, where u_{min} denotes the minimum lag distance of the tyre in mm during normal operation of the kiln. The ratio of circumferential lag to clearance is generally between 1.5 and 2.5.

Various tyre mounting systems are shown in Fig. 8.

1.2.4 Rotary kiln drive

The drive system comprises the two-piece girth gear (toothed ring), encircling the skiln shell, and the pinions (dual pinions for large, single pinion for small kilns), together with couplings, clutches, main and auxiliary gear units and drive motors. The kiln drive should be able to meet the requirements of all operating situations, including extreme cases (Figs. 9 and 10).

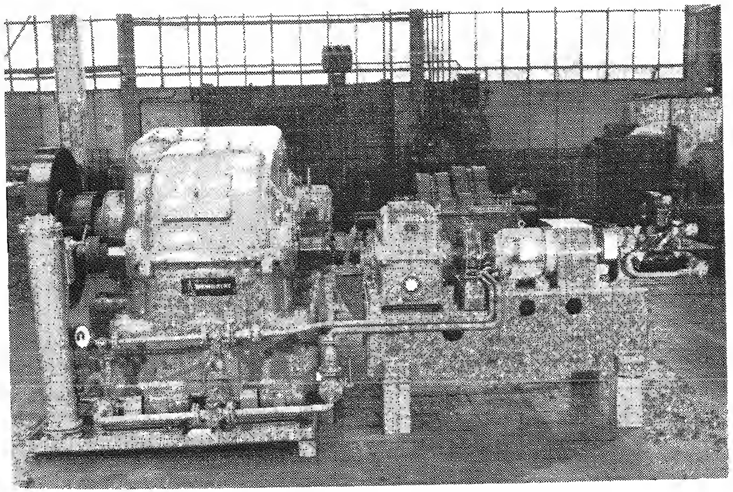


Fig. 9: Rotary kiln drive assembly (KHD Humboldt Wedag AG, Cologne)

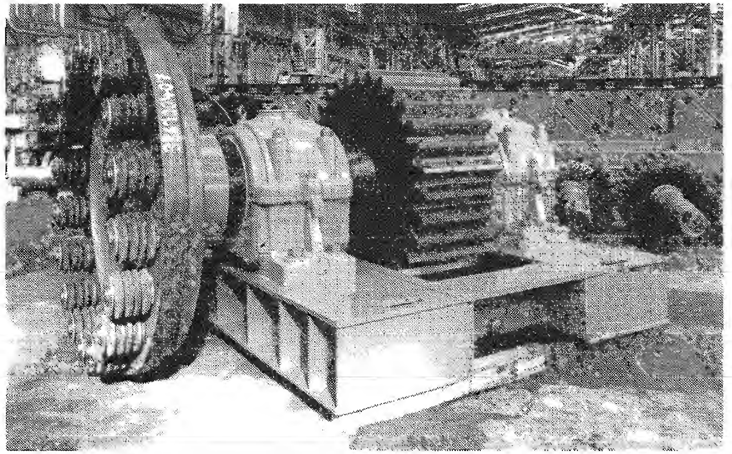


Fig. 10: Kiln drive pinion and mounting (KHD Humboldt Wedag AG, Cologne)

The girth gear and pinion(s) are accommodated in an oil-tight and dust-tight sheet-steel casing. Scoop lubrication is employed on small kilns, large ones are equipped with atomized spray lubrication.
Main drive motor. variable-speed thyristor-fed DC motor designed to a rating about 100% above the theoretical power demand.
Auxiliary motor: its purpose is to serve as a standby to enable the kiln to continue rotating (at reduced speed) in the event of a power supply failure or fault in the main motor. If the cement works has an emergency power supply system, the auxiliary motor may be a three-phase AC machine, otherwise an internal combustion engine – diesel or petrol (gasoline) – designed for quick starting will be provided.
Instead of girth gear drives, oil-hydraulic drive systems have occasionally been used for rotary kilns, but have not gained wide acceptance.

1.2.5 Air seals at kiln ends

For reasons of thermal economy it is necessary to prevent as effectively as possible the infiltration of ambient air into the rotary kiln at the feed (or inlet) and at the discharge (or outlet) end respectively. Specially designed seals are used which have to withstand high temperatures and also the wear caused by the abrasive dust contained in the kiln gases. Various forms of construction are employed. Figs. 11 and 12 show examples. An important requirement is that the seals should continue

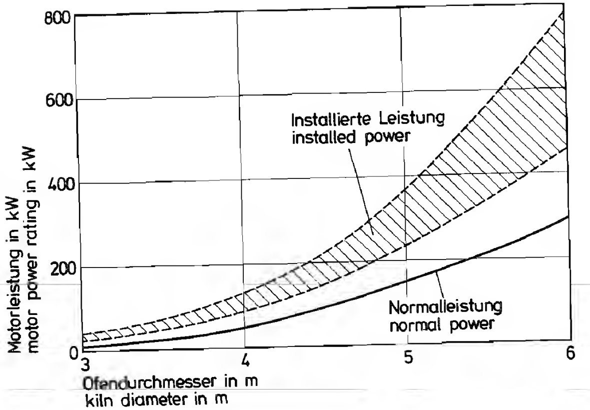


Fig. 11: Approximate drive power ratings for rotary kilns with cyclone preheaters

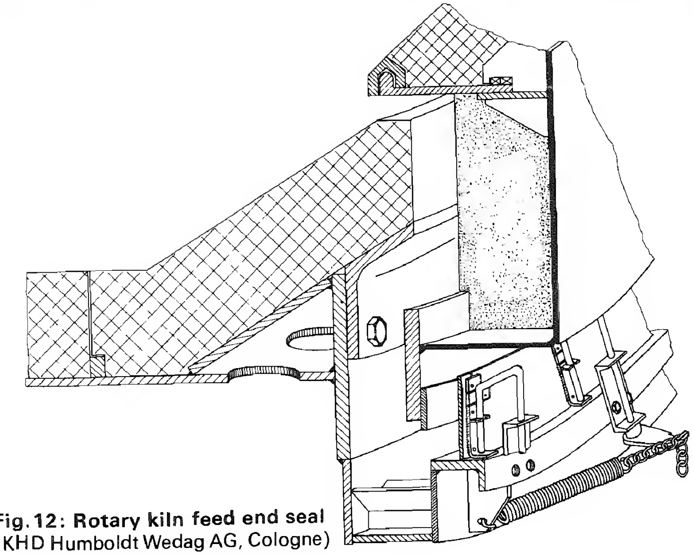


Fig. 12: Rotary kiln feed end seal (KHD Humboldt Wedag AG, Cologne)

to function properly in preventing air entry when they have undergone a certain amount of unavoidable wear and also if the kiln runs somewhat out of true. They must compensate for thermal expansion.

Careful maintenance of the kiln seals is important. Infiltration of air is liable to cause major heat losses. The amount of infiltrated air can be approximately estimated with the aid of the following formula:

$\mathring{V} = 30\,000\,A \cdot p$ (m^3/hour)

where A is the area of the gap (in m^2) and p is the negative pressure (suction) in the kiln (in mbar).

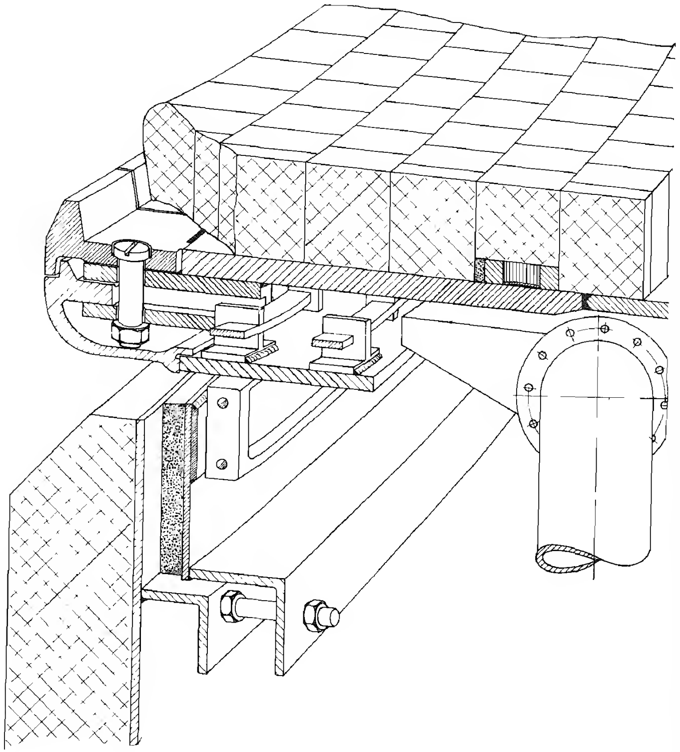


Fig. 13: Rotary kiln outlet end seal (KHD Humboldt Wedag AG, Cologne)

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2 Preheaters and precalcining

By E. Steinbiss

2.1 General

It was recognized quite early that the heat liberated in the rotary kiln can, besides being used for the actual processing of the feed material, advantageously be utilized for preheating the material. To promote this, the inlet zone (preheating zone) of the kiln may be equipped with suspended chains and/or inserts made of heat-resisting steel or ceramic materials which assist heat exchange (quadrantal inserts, etc.). Other devices serving the same purpose include, for example, sheet-steel compartments containing heat-exchanging media disposed around the circumference of the shell.

All such devices aim to provide a large contact surface area between the hot gases and the kiln feed material in order to promote heat exchange. If a kiln is fed with dry raw meal, these internal fittings stir up a great deal of dust which is swept along with the exit gases discharged from the kiln. To collect this dust a cyclone is interposed in the gas flow and can also usefully serve as a simple heat exchanger. From this principle were evolved more sophisticated heat exchangers with a view to further improving the thermal efficiency of the kiln system. These devices are external to the actual rotary kiln and installed at the feed end, where the hot exit gases flow through them and preheat the feed material and, if necessary, dry it. For the wet process of cement manufacture various types of slurry drying and preheating devices were developed.

With the dry process, fed with substantially dry pulverized materials (raw meal), the drying function is unimportant; what is important is the preheating attainable in suitable devices. Since a considerable part of the thermal processing of the material is accomplished in these heat exchangers external to the kiln, the kiln itself can be made correspondingly shorter.

2.2 Grate preheaters

Efforts to achieve further heat saving, i.e., improving the thermal efficiency of a kiln plant, led to the development of grate preheaters (Fig. 1 a, b).

More particularly, a preheater of this kind consists of a travelling grate carrying a bed of pellets (or nodules) formed from moistened raw meal in a pelletizing device.

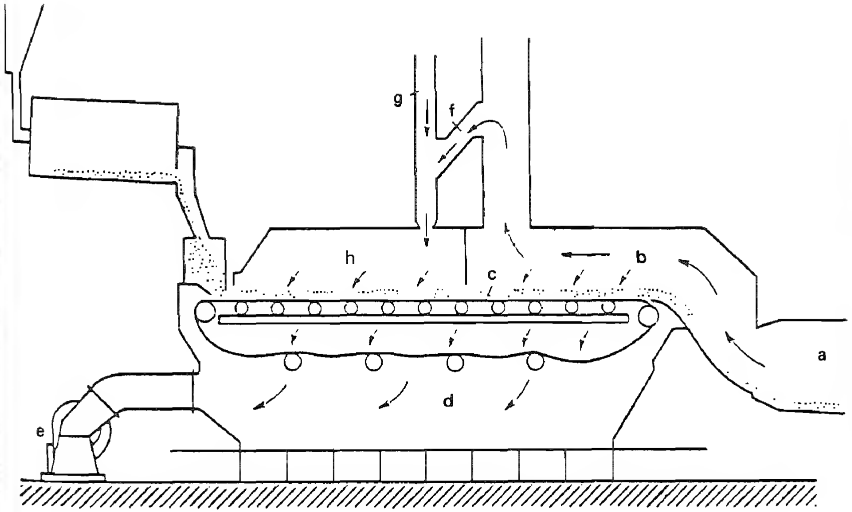

Fig. 1 a: Grate preheater (a rotary kiln, b hot chamber, c grate, d suction chamber, e fan, f overflow duct, g fresh air intake, h drying chamber)

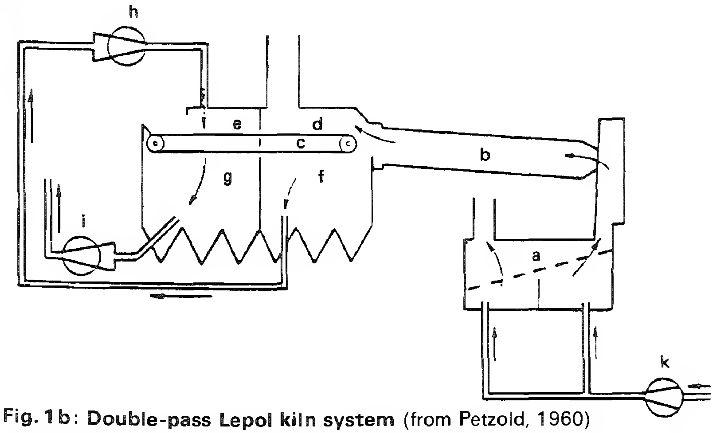

Fig. 1 b: Double-pass Lepol kiln system (from Petzold, 1960)
(a Fuller cooler, b rotary kiln, c Lepol grate, d hot chamber, e drying chamber, f + g suction chambers, h + i + k fans)

This method of cement manufacture is called the Lepol process, the preheater being known as the Lepol grate. The name is derived from that of the inventor, Dr. Lellep, and from Polysius, the firm that built the first grates. The hot kiln gases flow through the approximately 15 to 20 cm deep bed of pellets on the grate, either on the single-pass principle or, more particularly in later versions of the system, the double-pass principle. This process can be used not only for dry raw materials, but also in cases where they can be prepared only by wet methods, i.e., as a slurry. In that case the slurry is first dewatered as much as possible by means of filter presses and then moulded into cylindrical "fingers" which break into nodules. These are processed on the travelling grate in the usual way.

With raw materials prepared dry (raw meal) the pellets are formed with a certain quantity of added water in a pelletizer, usually in the form of a tilted rotating pan or dish, though drum-type pelletizing (or nodulizing) devices are still used to some extent. The pellets must have sufficient mechanical strength so as not to shatter when deposited onto the Lepol grate from the pelletizer. Moreover, they must possess a certain amount of plasticity to prevent them from prematurely disintegrating as a result of the relatively rapid heating they undergo on exposure to the hot kiln gases. Otherwise the fragments of broken pellets are liable to cause choking of the grate and thus obstruct the flow of gas through the bed, giving rise to serious trouble in operating the kiln.

2.3 Cyclone preheaters

The first application for a patent in respect of a cyclone preheater for raw meal was filed in Czechoslovakia by Vogel-Jörgensen, then employed by the Danish firm of F. L. Smidth. In due course the patent was granted, in 1934. It proposed preheating the raw meal in a cyclone separator before feeding it to the rotary kiln, the latter being correspondingly shortened in comparison with a conventional dry-process kiln. It was not till nineteen years later (1953), however, that the first functionally satisfactory cyclone preheater (or suspension preheater) was commissioned – for a 300 t/day guaranteed clinker output from a kiln in the works of Bomke & Blechmann, Beckum, Germany – after the technical practicability of the new method had been conclusively proved by Franz Müller. This first kiln plant with cyclone preheater was built by the firm of Humboldt (now KHD Humboldt Wedag AG), Cologne. Up to 1959 that firm was the sole supplier of the cyclone preheater, the prototype of which it had developed (Fig. 2). From then onwards, however, other German and foreign cement machinery manufacturers entered the market with their own versions of the cyclone preheater, all utilizing the same fundamental principle.

Whereas in 1953 most cement kilns had clinker outputs of between 300 and 500 t/day, nowadays kiln plants producing around 5000 t/day are not uncommon and are likewise equipped with cyclone preheaters embodying the same basic idea of heat transfer from the hot kiln exit gas to the raw meal in suspension in the gas stream. For further information the reader is referred to the article by Bomke (1978), which moreover contains a comprehensive list of literature references.

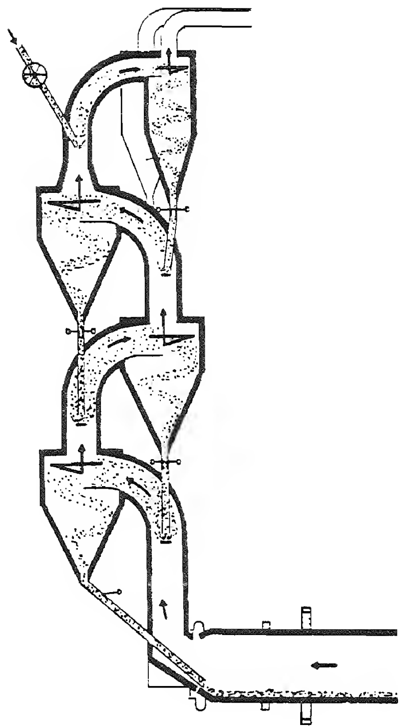

Fig. 2: Humboldt preheater
(from Bomke, 1978)

The demand for increasingly high clinker outputs from individual kiln plants resulted in corresponding increases in the size of the kiln shells and of the preheater cyclones. This general growth in the dimensions of the installations was attended by a number of problems and difficulties which had to be overcome. For example, it became impracticable to transport very large prefabricated kiln sections from the manufacturing works to the site of erection, the thermal rating of the burning zone in the kiln became very high and thus severely reduced the working life of the refractory lining, etc. These and other problems prompted cement plant manufacturers to consider the possibility of shifting more of the processing treatment from the actual kiln to the preheater, i.e., making the latter contribute more to the cement burning process than just preheating the raw meal. This approach led to the development of precalcining.

2.4 Precalcining processes

The precalcining principle and its applications have been developed more particularly in Japan and Europe. A feature which all precalcining systems have in common is that the supply of fuel is divided between two firing units, i.e., two burners or sets of burners, one in the kiln and the other in the suspension preheater. This principle is shown schematically in Fig. 3. The burners in, or associated with, the preheater are fed with combustion air consisting of exhaust air from the clinker cooler. This air is drawn either through the kiln itself or through a separate duct (the tertiary air duct). If the precalcining combustion air is drawn through the kiln, the latter has to be about 20% larger than if a separate duct is provided, and the air excess factor of the firing process is increased from 1.1 to about 2.1. The kiln volume rating (loading per unit volume) is about 2 t/m³ · day in conventional kilns, but in kilns with precalcining it is about 3.3 t/m³ · day if the precalcining combustion air is fed through the kiln and about 4 t/m³ · day if a separate tertiary air duct is provided. This means that in the last-mentioned kiln roughly twice as much clinker for a given internal volume of the kiln can be burned as in a conventional kiln without precalcining. At the same time, despite the much increased volume rating, the cross-sectional thermal rating is lower than that of the conventional kiln, the reason being that a much lower proportion (about 40%) of the total fuel supplied to the burning plant is actually fired in the kiln, the remainder being fired in the precalcining system.

As already stated, up to 60% of the fuel may be fired in the (pre)calciner. (With this proportion of fuel, the term "calciner" is perhaps preferable to "precalciner", since

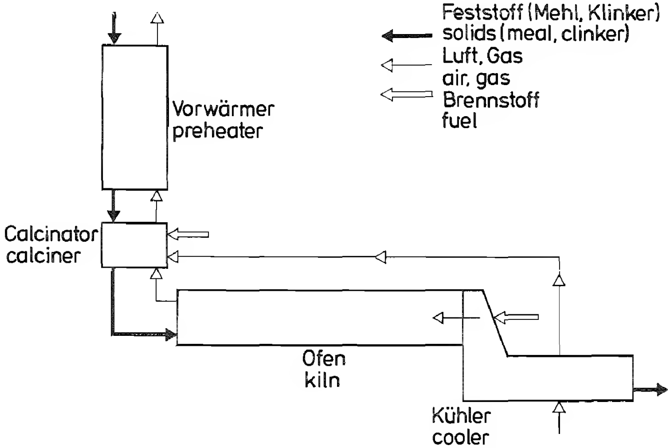

Fig. 3: Diagram illustrating the precalcining principle

the decarbonation of calcium carbonate is very largely accomplished in this device; on the other hand, the designation "precalcining" or "precalcination" is well established.) This process is especially advantageous when relatively low-grade fuels with low calorific value and/or high content of inert matter have to be used (charcoal, lignite, waste materials such as old motor tyres, etc.), as these can be fired in the calciner, where flameless combustion at relatively low temperatures below 900° C will suffice for obtaining the required calcination. Thus if 60% of the total fuel input is fired in the calciner, the raw meal will be about 90% calcined by the time it enters the kiln. With precalcining there is only a slight increase in heat consumption and a small rise in exit gas temperature as compared with the conventional kiln-cum-preheater system.

Precalcining with tertiary air supply through a separate duct is especially advantageous in conjunction with a bypass system for reducing the alkali content in the clinker. More particularly, with this precalcining equipment the heat losses associated with bypassing some of the kiln gas in order to reduce the so-called alkali cycle can be substantially cut down, more particularly because the undesirable constituents (alkalis, chlorides) are now volatilized in the kiln rather than in the preheater, so that a higher proportion of them can be discharged via the bypass with an equal amount of gas.

In a conventional preheater the raw meal is calcined only to a fairly limited extent (ranging from about 10 to 50%), the remainder of the carbon dioxide being driven out in the kiln itself. With precalcining almost the entire decarbonation process is effected in the calciner, one result of which is that the thermal conditions to which the refractory lining in the kiln is exposed become much less severe. This in turn means that, for equal clinker output, the diameter and length of the kiln can be correspondingly reduced. The significant feature is that the calciner is separate from the kiln and that part of the thermal energy required for the clinker manufacturing process is utilized in the calciner and not in the kiln. Besides, the heat contained in the exhaust air from the clinker cooler is also utilized (as preheated tertiary combustion air for the precalcining burners).

By conversion to precalcining, the clinker output of existing rotary kilns with cyclone preheater equipment can be substantially increased – by up to about 100% in certain cases. In new kiln plants equipped with (pre)calciners it is possible to increase clinker output up to threefold, as compared with a conventional kiln-cum-preheater plant, while the kiln dimensions (diameter and length) can moreover be reduced.

Further information on these and other aspects of precalcining systems will be found in the following bibliographic references.

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The occurrence of alkali and chlorine cycles in kiln systems and the possibility of controlling these phenomena by bypassing have already been mentioned. In general, these form part of the "dust cycle" problems which, in some plants may cause serious operational difficulties, such as may also arise from excessive formation of coatings in kilns. To find effective ways and means of coping with these problems has long claimed the attention of cement plant designers.

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3 Clinker cooling

By H. Xeller

3.1 Introduction: main types of coolers

The hot cement clinker discharged from the kiln is further treated in clinker coolers.

What all types of cooler have in common is that the cooling air flows directly – in counter-current or cross-current – through the clinker and that some or all of the heated air from the cooler is fed as combustion air to the kiln.
Water as a direct cooling medium for clinker is used only in the manufacture of special types of clinker or for after-cooling subsequent to cooling with air.
Indirect air cooling, with separating walls dividing the clinker from the air flow passages, is sometimes used, but only for after-cooling.
The main types of clinker coolers, listed in the order of their frequency of application, are:

direct coolers (Fig. 1);

grate cooler, planetary cooler, rotary cooler, shaft cooler;

indirect coolers, after-coolers (Fig. 2);

gravity cooler ("g" cooler).

3.2 Selection criteria and principal characteristics of coolers

The following aspects have to be considered in choosing the appropriate type of cooler in any given case:

raw material situation;
projected or existing kiln plant;
projected or existing works installations;
local environmental conditions.

The relative importance of the following requirements applicable to clinker coolers must be assessed accordingly:

obtaining good clinker quality by optimum cooling rate;

final cooling of the clinker to the lowest possible temperature;

optimum adaptation to the raw material drying system and burning system preceding the cooler;

least possible impact on the environment;

low capital cost;

low operating expenses,
i.e., favourable energy balance with a high proportion of heat recovery,
low electric energy consumption,
low wear and maintenance costs,
low susceptibility to faults (minimum downtime).

3.2.1 Clinker quality

The soundness, chemical resistance and strength of the cement, as well as the grindability of the clinker, are affected by the rate of cooling applied to the clinker.
The differences in cooling rate in the significant temperature range and for the commonly employed raw material compositions between the various clinker

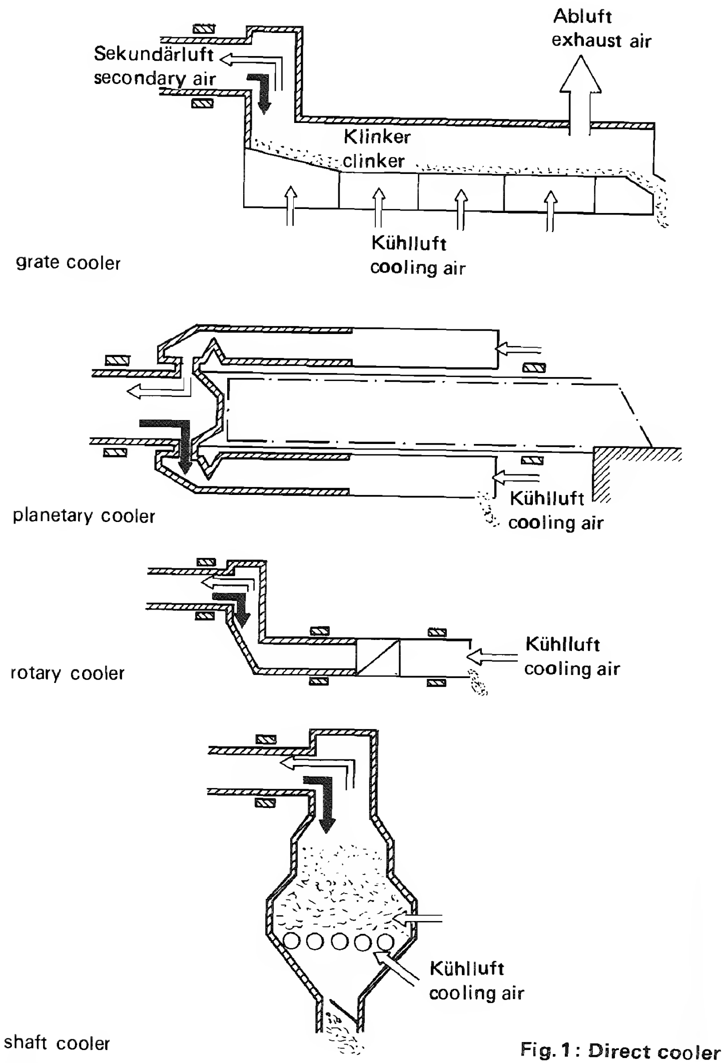


Fig. 1: **Direct cooler**

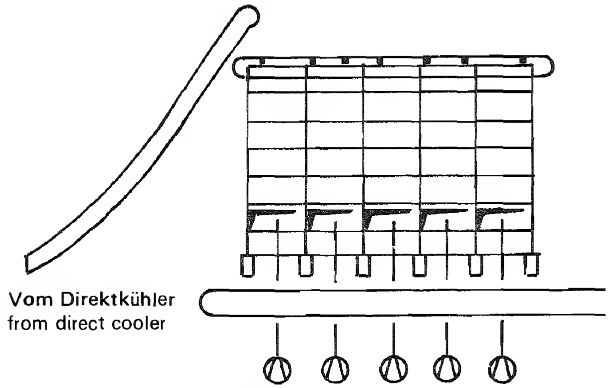


Fig. 2: **Indirect cooler, after-cooler**

cooler systems are, however, so small that for practical purposes there are no differences in the quality of the clinker finally obtained. Only the grindability of the clinker from grate coolers is, for equal grinding conditions, a little more favourable than that from other types of cooler.

3.2.2 Final cooling

Cooling of the clinker with air as the sole cooling medium cannot achieve so low a final temperature in the planetary, rotary or shaft cooler – in which all the cooling air has to be used as combustion air supplied to the kiln – as in the grate cooler. With the latter, especially if a clinker breaker is interposed, final temperatures of as low as 80° C for the cooled clinker can easily be attained, whereas the corresponding temperatures for rotary and planetary coolers, particularly if they are large ones, are generally above 150° C, while the clinker discharged from shaft coolers has temperatures above 300° C.
With these last-mentioned types of cooler the only way to attain lower final temperatures is by after-cooling or, with rotary and planetary coolers, alternatively by supplementary cooling with water.
Against this, cooling to low final temperatures in the grate cooler requires large quantities of cooling air. If the air heated in the cooling operation cannot be utilized for raw material drying, its dedusting before discharge into the atmosphere will involve heavy expenditure on dust collection equipment. For this reason it may even with grate coolers in certain cases be advantageous to employ separate after-coolers.

3.2.3 Scope for adaptation to the drying and burning system and to raw material conditions

The widest-ranging possibilities for adaptation to the requirements of economical drying of raw material and fuel, or the preheating of fuel, are afforded by the grate cooler. The various grate cooler designs range from systems with no exhaust air (for example, short grate coolers with additional after-coolers) to multiple-stage grate coolers embodying the duotherm principle with intermediate clinker breaking and intermediate air offtake.

With these arrangements, in cases where the raw material has a low moisture content and requires little exhaust air from the cooler for drying, additional expenditure on dust collection equipment for cleaning the exhaust air can be cut down. At the other extreme, raw material with as much as 14% moisture content can be dried without any extra heat input by utilizing the exhaust gas from the clinker cooler in combination with exit gas from a preheater kiln with precalcining.

With regard to adaptation to the kiln system the grate cooler is more versatile than the other systems of clinker cooler. More particularly, it offers the best conditions for optimal extraction of hot air for precalcining kiln systems with separate tertiary air supply.

For kiln systems with grate-type preheaters, in which the exit gases cannot be utilized for material drying purposes, there is practically no economical alternative to the grate cooler, because in such cases the exhaust air from the cooler can always be utilized.

On the other hand, no surplus hot air that can be used for material drying is available from planetary, rotary and shaft coolers. With the planetary cooler there is no possibility at all of obtaining tertiary air, while in the case of the rotary cooler and the shaft cooler a tertiary air offtake is indeed possible near the kiln hood or from the cooler shaft, but not without practical difficulties. See Table 1.

Table 1: Suitability of the various types of coolers for economical extraction of hot air

	grate cooler	planetary cooler	rotary cooler	shaft cooler
secondary air	×	×	×	×
tertiary air	×	–	0	0
hot air for coal drying	×	–	–	–
hot air for raw material drying	×	–	–	–

× = possible 0 = partly possible – = not possible

The chemical and mineralogical composition of the raw material is also of considerable influence on the effectiveness of the various cooling systems. For raw material which produces a very fine-grained clinker or gives rise to frequent dislodgment of coating in the kiln the best scope for adaptation to the burning system is provided by the grate cooler.

With planetary, rotary and shaft coolers the fluctuating rate of clinker discharge from the kiln due to ring formation and coating movements causes high and markedly varying final clinker temperatures, since the radiation heat losses remain substantially constant and the cooling air rate available to these coolers can practically not be altered.

With unequal granulometric and discharge conditions there are likely to be difficulties in operation more particularly in the shaft cooler.

The occurrence of large quantities of fine-grained clinker causes problems with all cooling systems, but grate coolers are best able to cope with such conditions because of the lower air velocities and less pronounced dust cycling effect in this type of cooler. See Fig. 3.

3.2.4 Environmental nuisance

A potential environmental nuisance due to dust and noise emissions is associated with grate coolers. With other types of coolers there is no discharge of dust, only the noise problem.

Dust emission:

The official clean air regulations in nearly all countries necessitate substantial extra expenditure on equipment for dedusting the exhaust air in cases where a conventional grate cooler is to be used. Centrifugal dust collectors with purely mechanical action are unable to meet the strict present-day requirements in this respect. Expensive granular bed filters, electrostatic precipitators or fabric filters have to be used for the purpose. For this reason, alternatives to the conventional grate cooler have been developed for use in cases where the exhaust air from the cooler cannot be utilized in raw material or coal drying installations. Examples of such modified grate coolers are the shortened grate cooler with ancillary after-cooler (the latter an indirect cooler with no dust emission) or the so-called duotherm grate cooler with intermediate indirect air-to-air cooler. Even so, the dust nuisance in the immediate vicinity of these modified grate coolers is still greater than that arising from rotary or planetary coolers, which have to be operated with a fairly high negative pressure at the rotating seal with the kiln hood, so that, because of the greater cycling effect, better retention of dust within the cooler system is obtained.

Noise emission:

All types of clinker cooler emit a great deal of noise, attaining levels of between 95 and 100 dB(A) at points of maximum loudness in the immediate vicinity (about 1 to 5 m distance). With grate and shaft coolers the cooling air fans are the principal

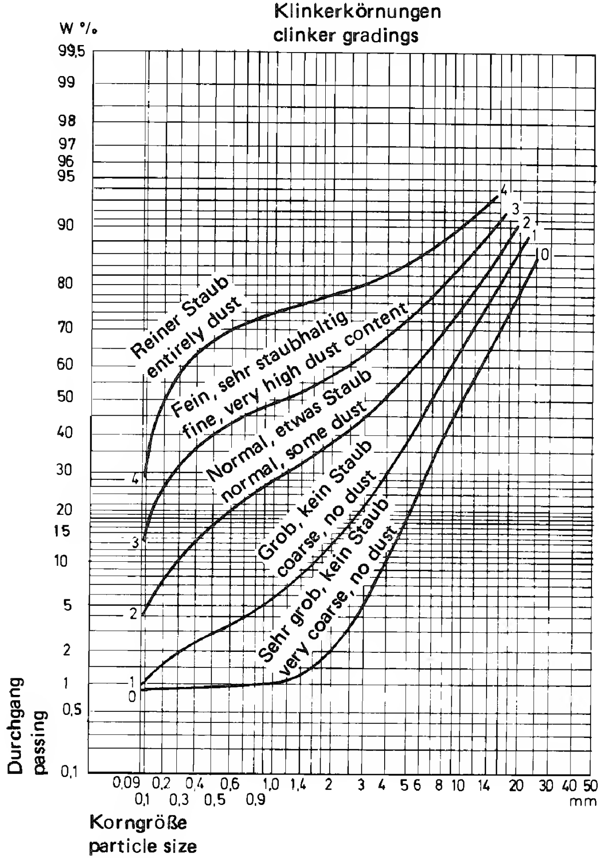


Fig. 3: Suitability of the coolers for different clinker gradings

Grate cooler: 1–3, for grading "0" only if equipped with intermediate breaker
Planetary cooler: 1 and 2
Rotary cooler: 1 and 2
Shaft cooler: 2

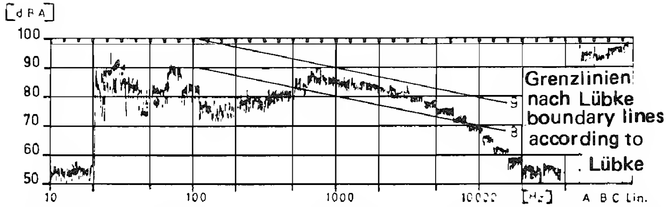


Fig. 4: One-third octave analysis of the sound emitted from a planetary cooler for normal kiln running at rated output (from Kadel, 1974)

noise emitters, whereas most of the noise from rotary and planetary coolers arises from the uninsulated lifter zone. See Fig. 4.
With all types of clinker cooler in locations susceptible to noise nuisance it is therefore necessary to apply noise control measures. Appropriate sound insulation arrangements are most elaborate and expensive in the case of planetary coolers because of the sheer size of the noise source, the elevated position thereof and the high ambient temperatures due to radiation and convection of heat. Depending on the distance from the cooler to adjacent residential areas, arrangements such as sound-attenuating walls, movable noise suppression covers or sometimes even totally closed buildings with forced ventilation may be necessary.

3.2.5 Capital cost

Besides the actual capital expenditure on machinery, electrical engineering components including measuring and control instrumentation, refractory and insulating material, buildings and erection of the cooler, the expenditure associated with the space requirements and the altitude (height above sea level) of the installation will also have to be considered.

Space requirements

A rotary cooler which is installed under the kiln and in the direction opposite to that of the material flow in the kiln represents an economical arrangement in terms of the space needed to accommodate it. An even more favourable arrangement in this respect is the grate cooler with complete exhaust air utilization, installed under the kiln. However, for operational reasons it is better not to place the clinker cooler with its material flow direction opposite to that in the kiln.
A shaft cooler requires very little extra space in the horizontal directions, but on account of its great headroom it may cause problems on sites with unfavourable soil conditions.
Planetary coolers and those grate coolers which have to operate in conjunction with highly efficient dust collection equipment (granular bed filters, electrostatic precipitators, fabric filters with air-to-air coolers) will require a relatively large

amount of space. On the other hand, planetary coolers require the least headroom and can therefore be advantageous on sites with critical ground-water conditions, i. e., waterlogged sites where subsurface construction (pits, chambers) presents difficulties.

Altitude

With increasing altitude of the site above sea level the density of the atmosphere becomes lower, so that the required volumes of cooling air and combustion air become larger. This most unfavourably affects rotary coolers, whose diameter is determined by the air velocity in the cooling tube and in which therefore, at higher altitude, the specific throughput is reduced while the capital cost of the cooler increases. Similar considerations apply to the planetary cooler, though in this type there are generally somewhat ampler reserves or margins with regard to the critical air velocity.
Grate coolers and shaft coolers present the least problems in this respect, because the increase in fan capacity and size of housing necessitated by higher altitude has only a minor effect on the overall cost of the cooler. On the other hand, with the shaft cooler the attainment of suitably low final clinker temperatures becomes even more problematical at high altitudes, while in the case of grate coolers it becomes advisable under such circumstances to install a type comprising an intermediate breaker and a circulating air system.
The actual capital expenditure associated with the various types of clinker cooler is liable to vary greatly from one case to another, especially if after-coolers and/or elaborate noise control measures have to be included. In every case where complete utilization of the exhaust air from the cooler is possible, the multiple-stage grate cooler with circulating air system will involve the lowest capital outlay.

Mechanical and ancillary equipment

Wherever complete exhaust air utilization is possible, the grate cooler will always be the least expensive type in terms of purely mechanical engineering and ancillaries. In this respect the shaft cooler is also quite favourable, whereas in the case of the rotary cooler and planetary cooler, as also the grate cooler with high-efficiency dust collection system, the cost of mechanical and/or ancillary equipment is distinctly higher, though in this there is very little difference between these three last-mentioned types of cooler.

Electrical equipment, instrumentation

In so far as these expenditure items are concerned, the planetary cooler is distinctly superior to the other types even though it requires a more powerful kiln drive and higher-capacity exit gas fan, while the shaft cooler and grate cooler are the most expensive types in this respect.

Refractory lining and insulation

The highest cost arises in planetary coolers, while grate coolers are least expensive.

Structural engineering

Planetary and rotary coolers are much less expensive in this respect than shaft and grate coolers.

Erection

Because of the more difficult conditions due to the handling of heavy parts and the elaborate welding work involved, planetary coolers are substantially more expensive to erect than the other types of cooler. The latter differ little from one another in erection costs.

3.2.6 Operating costs

The operating costs of clinker coolers mainly comprise the direct expenditure on:

replacement of wearing parts, repair materials,

wages for repairs and maintenance;

energy costs comprising electricity and heat (the latter because heat recovery is never 100%).

Besides, indirect expenditure has to be taken into account. This arises when, as result of faults or deficient operation of the clinker cooler, the plant is unable to run at optimum efficiency or indeed has to be shut down.
The relative operating cost items are very similar for the various types of cooler. The cost relationships exemplified by a grate cooler with exhaust air utilization are indicated in the accompanying diagram (Table 2). These percentage figures are based on the following assumptions:

cost of heat:	(21 DM/10^6 kcal/kg) 5 DM/GJ
electricity:	0.075 DM/kWh
repair wages:	17 DM/h
repair materials:	see Table 2.

The high proportion of expenditure on energy as compared with that on repairs and parts clearly emerges from the diagram.

3.2.6.1 Heat input and heat recovery

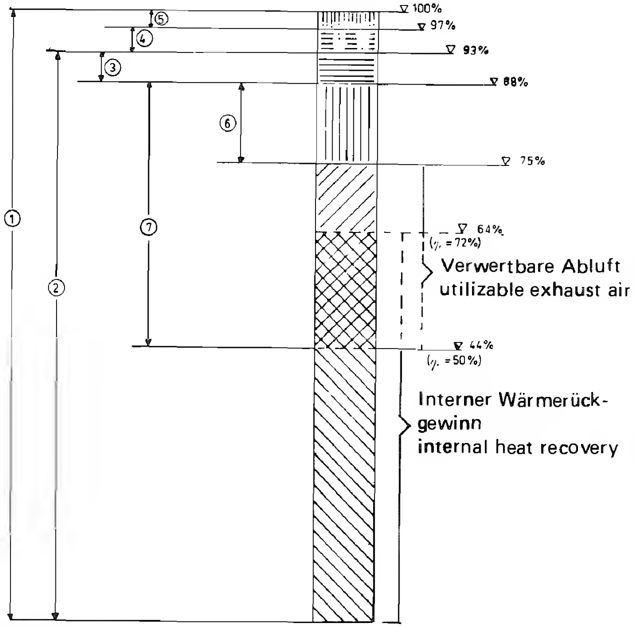
The cost of heat for the cooler is taken to comprise all expenses attributable to that proportion of the clinker heat which is not recovered, i.e., not utilized in one of the following possible ways:

- preheating the combustion air for firing the kiln;

- utilizing the heat in the exhaust air from the cooler for materials drying or other heating purposes outside the burning system (coal, raw material or slag drying; preheating of fuel oil or water).

The proportion of heat recovery is sometimes referred to as the thermal efficiency of the cooler. More particularly this denotes that proportion of the total heat content of the clinker (as it is discharged from the kiln) which is utilized for one or more of the above-mentioned purposes. A distinction may further be drawn between the

Table 2: Operating cost relations for the grate cooler with exhaust air utilization



1) overall cost (if no heat recovery at all)
2) cost of energy (if no heat recovery at all)
3) cost of power
4) replacements for wear and repairs
5) wages for repairs and maintenance
6) cost of heat in the case of optimum heat recovery (non-utilizable thermal radiation and convection, final temperature of clinker, water)
7) cost of heat in the case of 50% internal heat recovery

internal thermal efficiency, which relates only to the heat utilized within the clinker burning process itself, and the external thermal efficiency, which takes account of the entire quantity of clinker heat that is utilized. This latter efficiency concept has significance only in the case of grate coolers from which the exhaust air can indeed be utilized for its heat content.

The thermal efficiency is calculated from:

$$\eta = \frac{Q_{cg} - Q_{cl}}{Q_{cg}} \times 100\ (\%)$$

where

Q_{gc} = heat gain of cooler = $Q_{cli} + Q_{air}$
Q_{cli} = heat content of clinker discharged from kiln
Q_{air} = heat content of cooling air
Q_{cl} = heat loss from cooler = $Q_{hc} + Q_{r+c} + Q_w + Q_{ex}{}^*$
Q_{hc} = heat content of clinker discharged from cooler
Q_{r+c} = heat loss due to radiation and convection
Q_w = heat loss due to water cooling (water injection, water-cooled plate)
$Q_{ex}{}^*$ = heat content of exhaust air from cooler (in calculating the external thermal efficiency only the proportion of unutilized exhaust air is considered).

The efficiency of the clinker cooler is governed not only by the type and design of the cooler, but also by the following:

clinker entry temperature;
secondary air flow rate,
granulometric composition of the clinker;
exhaust air heat not utilized in the burning process.

These last-mentioned four factors are dependent on the kiln and the material conditions, not on the design and manner of operation of the cooler. For this reason it is necessary to exercise due caution in making comparisons between coolers on the basis of thermal efficiency.
For example, the internal efficiency of a cooler will decrease if the secondary air rate is reduced and/or the clinker entry temperature is lowered. Both these quantities are largely dependent on factors outside the influence of the cooler.
The secondary air rate is determined by

the heat consumption of the kiln plant as a whole;
the primary air rate,
the amount of inleakage of air at the kiln hood,
the air excess with which the kiln is operated.

The clinker entry temperature, i.e., the temperature at which it is discharged from the kiln and enters the cooler, depends more particularly on the length of the flame and of the burner in the kiln. If the firing nozzle is fairly long, part of the kiln will in

effect function as a cooling zone. As a result, the cooler efficiency will be less good, but the overall heat consumption of the burning plant will in general be somewhat improved. A long firing nozzle has the additional advantage that the thermal rating – the "heat load" or thermal intensity per unit area of wall surface or unit volume of internal space – of the kiln outlet and of the cooler is reduced. This is an important advantage more particularly in kilns equipped with planetary coolers, while in grate coolers it especially reduces the formation of objectionable accretions ("stalagmites" or "snowmen") in the chute or shaft leading into the cooler. Instead, a clinker dust-ring is, in such cases, likely to build up in the kiln itself, but such rings generally do not grow beyond a certain size, after which they collapse spontaneously and break up.

The secondary air rate (L_{sec}) can be calculated as follows:

$L_{sec} = L_{com} - L_{pr} - L_{inf}$ (Nm^3/kg clinker)

where: L_{com} = combustion air rate
L_{pr} = primary air rate
L_{inf} = rate of air infiltration (inleakage) at kiln hood

and:
$L_{com} = n \cdot L_{min} \cdot K$ (Nm^3/kg clinker)
where: n = air excess factor
L_{min} = minimum quantity of air required for complete combustion and dependent on the heat consumption of kiln and type of fuel used
K = rate fuel to clinker (kg/kg)

Since L_{min} is dependent on the fuel fired in the kiln, it will have to be calculated from the elementary analysis thereof for each individual case.

The following approximate values may be adopted for the standard fuels of average composition (H_u = net calorific value):

coal: $$L_{min} = \frac{1.001}{1000} H_u + 0.5505 \ (Nm^3/kg)$$

(corresponding to about 1.08 Nm^3/1000 kcal)

heavy fuel oil: $$L_{min} = \frac{1.228}{1000} H_u - 1.37 \ (Nm^3/kg)$$

(corresponding to about 1.089 Nm^3/1000 kcal).

natural gas: L_{min} = approx. 1.092 Nm^3/1000 kcal.

The air excess can be calculated from the exit gas analysis at the feed end of the kiln as follows:

$$n = \frac{1}{1 - 3.762\ (O_2 - 0.5\ CO)/N_2}.$$

With a favourably designed kiln burner the following primary air rates will be required:

coal:	approx	7–12%	of combustion air rate
heavy fuel oil:	approx.	3– 5%	of combustion air rate
natural gas:	approx.	0– 3%	of combustion air rate.

From these figures it is evident that, for example, when the fuel is changed from coal to natural gas, the calculated efficiency of the cooler becomes higher, even if equal clinker cooling rate curves are assumed in both cases. The reason for this increase in efficiency is that the secondary air rate for natural gas firing is higher than for coal firing. Yet, as a result of this change-over of fuel, the heat consumption of the plant as a whole will increase for other reasons (higher exit gas rate, poorer heat transfer from the flame).

The proportion of infiltration air at the firing hood of the kiln will depend on the design and condition of the seal. The magnitude of the negative pressure in that part of the system is also of major importance with regard to this. The greatest negative pressures occur in the hoods of kilns with planetary or rotary coolers, because with these cooler systems the air flow resistance in the cooler has to be overcome by the kiln fan. In the case of the rotary coolers, in particular, the entry of infiltrated air can be a problem, because with this type of cooler, besides the high negative pressure required, there are two rotating seals where inleakage of air may occur.

The secondary air requirements of wet-process kilns or long dry-process kilns with a specific heat consumption of 5.0–5.5 GJ/t of clinker (1200–1300 kcal/t) is in the region of 1.3–1.5 Nm^3/kg of clinker.

Heat-saving kilns with less than 3.3 GJ heat consumption per t of clinker (790 kcal/kg) require secondary air at a rate of about 0.85–0.9 Nm^3/kg of clinker.

Because of the considerable effect that the secondary rate has upon the clinker entry temperature and the calculated efficiency of the cooler, the relevant values are, for the sake of better comparability, sometimes converted to equal secondary air rate and, with the aid of the cooling curves determined, also to equal clinker entry temperature. However, this procedure is meaningful only in those rare cases where the granulometric characteristics of the clinker in the respective plants to be compared are also similar.

The heat losses assignable to the following items are indicated in Table 3·

sensible heat in the clinker leaving the cooler;
radiation and convection;
water cooling;
exhaust air;
secondary air.

The next two diagrams (Figs. 5 and 6) show the various loss proportions – varying with the clinker exit temperature – for grate coolers and for planetary or rotary coolers, on the assumption that these operate with a heat-saving kiln system and that these coolers can at best attain about 66% efficiency.

Table 3: Heat balances of coolers in GJ/t (kcal/kg)

type of cooler	grate coolers					tubular coolers		shaft cooler
	travelling grate	inclined grate	horizontal grate	com-bination	multistage duotherm	planetary	rotary	
heat supplied								
clinker	1.507 (360)	1.507 (360)	1.507 (360)	1.507 (360)	1.507 (360)	1.222 (292)	1.356 (324)	1.507 (360)
cooling air	0	0	0	0	0	0	0	0
clinker dust	–	–	–	–	–	(0.096 (23)	0.108 (26)	–
heat expenditure								
clinker	0.092 (22)	0.067 (16)	0.079 (19)	0.067 (16)	0.033 (8)	0.117 (28)	0.158 (38)	0.301 (72)
secondary air	1.009 (241)	1.026 (245)	0.950 (227)	1.026 (245)	1.080 (258)	0.896 (214)	1.005 (240)	1.194 (285)
clinker dust	–	–	–	–	–	0.054 (13)	0.067 (16)	–
radiation and convection	0.017 (4)	0.017 (4)	0.017 (4)	0.017 (4)	0.025 (6)	0.251 (60)	0.209 (50)	0.012 (3)
exhaust air	0.364 (87)	0.397 (95)	0.461 (110)	0.397 (95)	0.369 (88)	–	–	–
water cooling	0.025 (6)	–	–	–	–	–	0.025 (6)	–
therm. efficiency								
internal %	67	68	63	68	72	72	73	79
external %	–	–	–	–	96	–	–	–

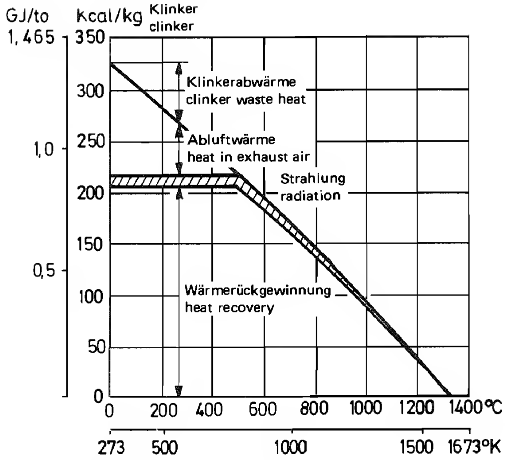


Fig. 5: Conditions for the grate cooler

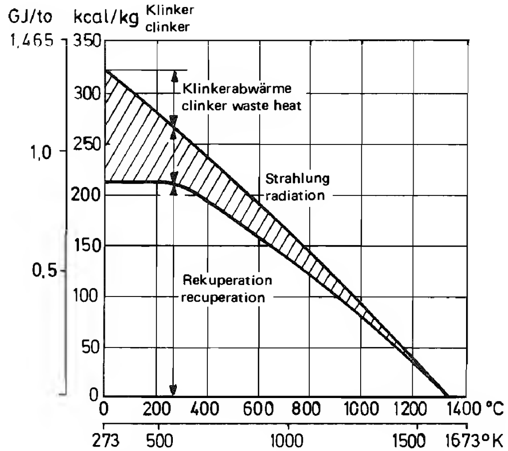
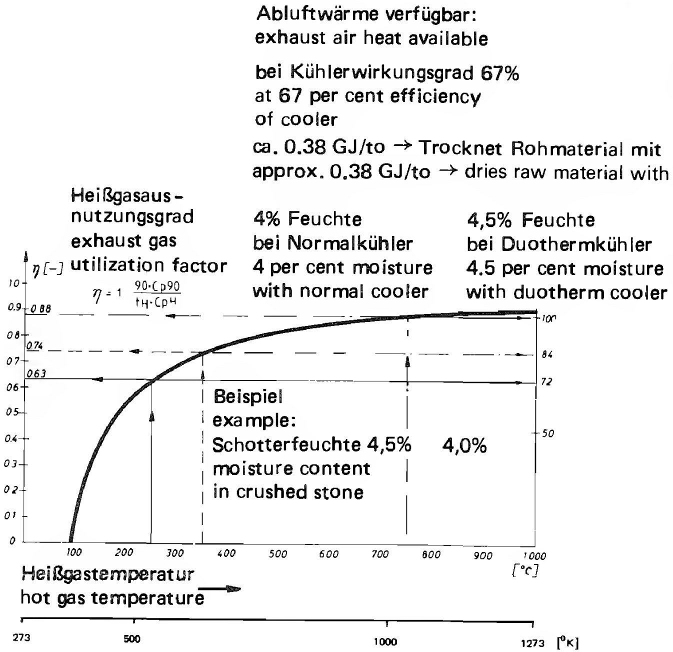


Fig. 6: Conditions for the rotary and planetary coolers; example shows respective proportions in a case where the final temperature of the clinker is about 250° C



Example: moisture content in crushed stone		4.5%		4.0%
		erforderliche Wärmemenge heat required		
750°C	Heißgas: hotgas:	0.33 GJ/to	Klinker clinker	0.27 GJ/to
350°C	Abluft: exhaust air:	0.39 GJ/to	Klinker clinker	0.32 GJ/to
250°C	Abluft: exhaust air:	0.46 GJ/to	Klinker clinker	0.38 GJ/to

Fig. 7: Fuller cooler — exhaust air utilization

The heat lost from the grate cooler in the exhaust air can in part be recovered. With **exhaust air utilization** the most favourable conditions are obtained if the largest possible thermal gradient is available for the kiln by the different types of coolers. The best hot gas utilization ratings for raw material drying are obtained with grate coolers equipped with air circulation systems.
The amount of heat given off to extraneous systems can permissibly by credited to the cooler only if this really does result in heat savings in those systems. This is illustrated in Fig. 7.
The amounts of hot air needed for drying the same raw material will differ greatly according to the temperature of the available hot air. For example, if the raw material has a moisture content of 4.0%, the utilization rating for hot air with a temperature of 350°C is 74% and the heat requirement is 0.324 GJ/t of clinker (77.5 kcal/kg), while if the hot air has a temperature of 250°C the corresponding figures are 63% and 0.379 GJ/t (90.5 kcal/kg).
From these data it also appears that with preheater kilns it is always most economical first to utilize the kiln exit gas before utilizing the exhaust air from the cooler.
Because of the intermediate clinker breaker and hot air return, the multiple-stage cooler with duotherm air circuit is characterized by high exhaust air temperatures in conjunction with good internal thermal efficiency. This type of cooler will therefore always constitute the most economical solution in cases where the raw material has a high moisture content and therefore needs considerable heat input for drying.
For low thermal gradients the heat utilization is economically limited that is especially for heating of water vapor or fuel oil. The heat recovery that can be achieved depends very greatly on the granulometric composition of the clinker. In the case of the planetary and the rotary cooler the efficiency will, with very fine-grained clinker, additionally be affected by the heat losses due to the dust cycle.

3.2.6.2 Electric energy requirements

Besides heat, the input of electric energy also comes into the energy balance of the clinker cooler. Table 4 reviews the average energy consumption figures of the various systems of cooler.
As appears from this table, the lowest demands are made by the planetary cooler. This type of cooler indeed does not directly consume any electric energy at all, but the energy consumption of the kiln drive is of course higher (because the cooler is rotated along with the kiln itself), as is also that of the exit gas fan drive. Most expensive in terms of electricity consumption is the shaft cooler, namely, about 6.7 kWh/t higher than the planetary cooler. Adopting the energy prices indicated in Section 2.6, this difference corresponds to 0.1 GJ/t of clinker (about 24 kcal/kg) or a difference in cooler efficiency of about 7%.

Table 4: Electric energy consumption of coolers

		grate coolers		planetary cooler	rotary cooler	shaft cooler
		normal type	multistage with duotherm system and intermediate breaker			
cooler (fans, drives)	kWh/t	3.4	4.5		3.5	8.5
exhaust air fan	kWh/t	1.8	1.5		–	–
proportion for kiln drive and exit gas fan	kWh/t	–	–	2.0	0.3	0.2
total	kWh/t	5.2	6.0	2.0	3.8	8.7

Table 5

material designation	chemical composition % Cr.	Ni.	other	max. service temp. °C	embrittlement	price DM/kg 1977	application
PG 6	6	–	–	600	–	4.5– 5	grate plates, cool part
4710	7		1 Mn	850	–	6– 8	scoops
4729	13	–	2.3 Si	900	–	5– 7	lifter flights
4777 (FM × 1430)	30		0.6 Mo	1100	+	6– 8	lifters, hot zone
4822	24	4.5	1.5 Mn	1100	+	8–10	lifters flights
4823	27	4.5		1150	+	6– 8	nose sectors
4835	25	8		1100	–	9–15	lifters, transition section
PG 2710	27	10		1100	–	7– 9	grate plates, hot zone
PG 2512	25	12		1100	–	7– 9	grate plates, hot zone
4832	20	14	1.8 Si	1000	–	10	lifters, transition section
FMR 71	35	20		1150	–	13	lifters, refr. lined zone
Umco 50						35–40	grate plates, hot zone

3.2.6.3 Costs associated with wear, repairs, materials and wages

Provided that coolers of well developed and proven design are employed, the cost arising from repairs and the replacement of wearing parts is relatively low in comparison with energy costs. As a rule, the effect of material conditions (granulometric composition of the clinker, movements of coating material, clinker discharge) is greater than that of the type or design of the cooler. In planetary and rotary coolers the main wearing parts are the lifter inserts, while in grate coolers they are the grate plates. The most economical design and construction features of these respective wearing parts will depend on the burning conditions and the nature of the clinker and will usually have to be determined by an empirical approach. The grades of material used for these parts are reviewed in Table 5.
For planetary and rotary coolers the choice of the refractory bricks and monolithic refractories is furthermore important and should receive due attention.

3.2.7 Availability – indirect expenditure

The effect of the material conditions on the operational availability of the cooler is generally more important than the design or type of cooler itself. With properly developed cooler designs it can be presumed with regard to all types of cooler, if they are given careful and methodical maintenance and if favourable material conditions exist, that the clinker burning plant will not be subject to any substantial downtime due to trouble with the cooler and that any repairs that become necessary can be carried out during the periodic shutdowns for relining the kiln.
From the viewpoint of constant readiness for service the rotary cooler has proved especially advantageous in practice. Damage to the inserts and to the refractory lining in such coolers are least serious in their consequences and least often necessitate plant shutdowns.
On the other hand, conditions for the planetary cooler are more critical. Another disadvantage of such coolers is that while relining work is being carried out in the kiln, the execution of repairs in the cooler is awkward because the kiln and cooler cannot be rotated independently of each other.

3.3 Description of the various types of clinker cooler

3.3.1 Grate coolers

Nearly all manufacturers of machinery for the cement industry include their own grate coolers in their product ranges. Hence it is not possible, within the present scope, to go beyond a description of the more commonly encountered types and forms of construction.
In the main, a distinction is drawn between travelling grate coolers and reciprocating grate coolers.

3.3.1.1 Travelling grate coolers

The Recupol cooler made by the firm of Polysius is a widely used example of the travelling grate cooler.
The grate consists of an endless "belt" or "chain" of grate elements and resembles the grate of a Lepol kiln. In the cooling process the clinker is at rest upon the travelling grate plates which, being in constant motion, are exposed to high temperatures only for short periods, so that very little wear on the plates occurs. Hence these can be made of spheroidal cast iron, a relatively inexpensive material. A further advantage is that any damaged plates can be replaced by new ones while the plant continues in operation (Fig. 8).

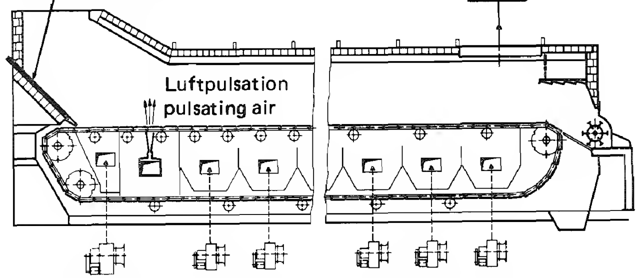


Fig. 8: Travelling grate cooler (from Herchenbach, 1978)

The rotational speed of the drive shaft of the cooler is variable. Grate travel speeds range between 0.5 and 2.5 m/minute, and the depth of the bed of material is between about 120 and 180 mm, depending on the grate speed. Average grate design loads are about 20–30 t of clinker per m^2 and per day.
The cooling is effected in two zones: the primary (or precooling) and the secondary (or final cooling) zone. In the primary zone pulsating air is blown into the bed of clinker through the grate slots, so that conditions resembling those in a fluidized bed are produced, resulting in a powerful cooling action. At the same time, this aeration helps to distribute the clinker uniformly across the width of the grate, with the coarser particles underneath and the finer ones at the top of the bed. A lower air pressure is employed in the secondary cooling zone, so that the clinker settles down at rest here.
Specific cooling air rates range from about 1.8 to 2.4 Nm^3/kg of clinker, achieving final clinker temperatures averaging 120°–150° C. As will be explained with reference to the reciprocating grate cooler later on, air recirculation can be applied with the travelling grate cooler, thus reducing the amounts of exhaust air discharged.

A clinker breaker (hammer crusher) extending across the full width of the grate reduces the larger clinker fragments and throws them back onto the grate for further cooling. A chain curtain protects the refractory lining.
The main problems associated with the travelling grate cooler are those of achieving uniform distribution of the clinker across the width of the grate immediately after its discharge from the kiln.
This distribution problem is not difficult to solve in small and medium-sized coolers operating with kilns fed with preformed pellets or nodules. The clinker falls onto a chute equipped with a water-cooled steel plate, the purpose of which is to prevent the formation of hot clinker stalagmites ("snowmen") at the inlet to the cooler. The slope of this plate can be adjusted, as also its position in the transverse direction, in order to obtain satisfactory distribution of the clinker.
In order to avoid having to use a cooling plate with its associated heat losses and in order to obtain evenly distributed clinker across the grate width also in larger coolers, the design of the front end of the cooler was modified as shown in Fig. 9. In this arrangement the foremost part of the grate (about 10% of its overall length) rises at an angle of 45 degrees, so that a transverse "trough" is formed at this end of the cooler. Cooling air is introduced here under high pressure, achieving uniform distribution of the clinker, while the grate extracts from this trough a bed of clinker which, it is claimed, is of constant depth over the full width of the grate.

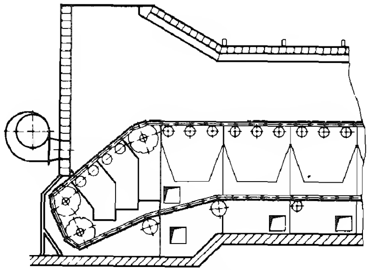


Fig. 9: Travelling grate cooler with rising grate (from Herchenbach, 1978)

3.3.1.2 Reciprocating grate coolers

Many variants of the reciprocating grate cooler are available from the cement machinery manufacturers. Most of them are, however, closely similar in principle to the most commonly employed type, namely, the Fuller cooler. This cooler and its many offshoots and variants comprise such types as the horizontal grate cooler, inclined grate cooler and combination cooler, with or without a gravity cooler ("g" cooler) for after-cooling, and multiple-stage coolers with intermediate clinker breaking. Furthermore, systems with or without air circulation are available. A feature shared by all these coolers is their cross-current and counter-current cooling action.

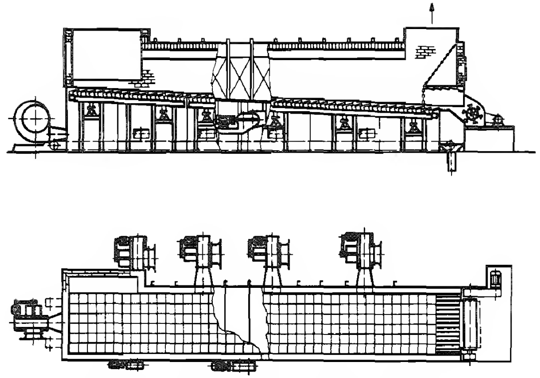


Fig. 10: Inclined grate cooler (from Herchenbach, 1978)

3.3.1.2.1 General design features

The reciprocating grate system (Fig. 10) comprises rows of alternately fixed and movable grate plates, secured by means of T-bolts to grate support girders. The plates are of various grades of steel along the cooler, corresponding to the different thermal and mechanical loading conditions. In the hottest part the plates are mostly of chrome-nickel alloy (see Table 5), while in the after-cooling part they can suitably be made of cast chrome steel. The grate plates are about 300 mm × 400 mm in size, while the length of stroke is 120 mm. Allowing for overlap, the effective length of a plate is 323 mm. The shape of the plates is suited to the requirements of the clinker bed. Thus, the following types are to be distinguished: tapered, flat and curb plates, also plates with and without holes. The holes in the rear parts of the plates acts as nozzles, directing the cooling air flow vertically upwards. Air is forced horizontally into the bed of clinker through the gaps between the fixed and the movable plate rows and through the holes in the end faces of the plates.
The two cooling air streams and the continual agitation of the clinker by the grate movements ensure that the clinker particles come into intimate contact with the air. Grate ratings (specific loads) in modern coolers are between 26 and 55 t of clinker per day and per m^2 of active grate surface area
For attaching the grate plates to the support girder the so-called fingerless construction is used, enabling the plates to be dismantled by withdrawing them downwards.
The fixed girders are mounted on bearings bolted to the side walls of the cooler
The movable girders are interconnected via a framed assembly, the so-called movable frame. The latter is carried by two or more shafts which in turn are

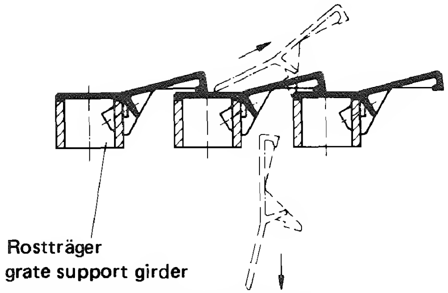


Fig. 11: Dismantling a clinker cooler grate plate

supported on wheels which run on short guideways. The bearings and drive of the shafts are located outside the windboxes, so as to facilitate testing and servicing while the cooler is in operation (Figs. 12 and 13).
The drive system either comprises a crank mechanism with P.I.V. variable-speed drive or variable-speed motor More recently, direct pneumatic drive has been introduced. The grate operating frequency ranges from 3 to 20 strokes per minute (Fig. 14).

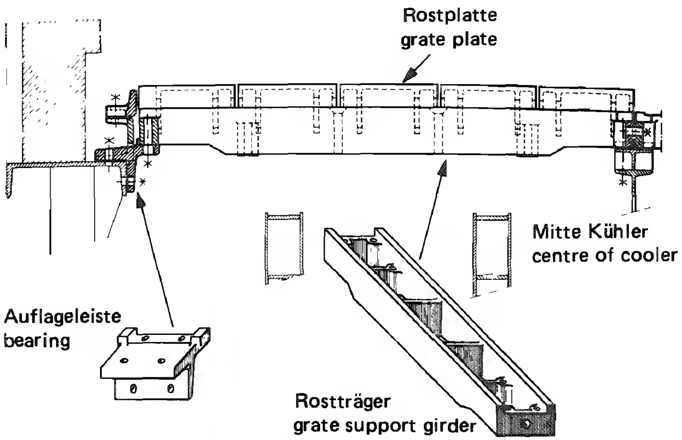


Fig. 12: Fixed grate support girder

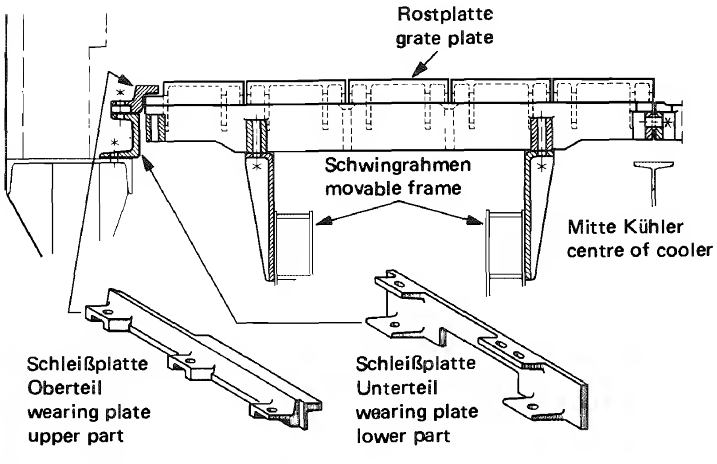


Fig. 13: Movable grate support girder

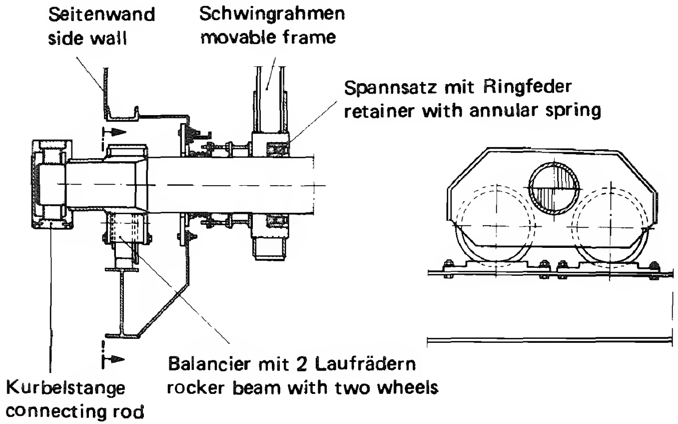


Fig. 14: Drive shaft of cooler (new type)

The grate is enclosed in a refractory-lined sheet-steel housing. The hot clinker discharged from the kiln falls through the inlet shaft directly onto the grate. In earlier designs the clinker first landed on a feed shelf and was then distributed on the grate. The drawback of this arrangement was the tendency for stalagmite formation ("snowmen"), which was counteracted by providing a water-cooled steel plate at the entrance to the cooler.
From the hot part of the cooler the clinker is shoved along towards the outlet end, while undergoing continuous agitation during its progress through the cooler. At the outlet most of the clinker falls through the grizzly (bar screen), while the oversize particles are fed to a hammer crusher. A chain curtain protects the walls and roof of the cooler against damage by pieces of clinker flung back by the crusher.
The casing over the grate is so amply dimensioned that the air flow velocities remain low, even if there is a deep bed of clinker on the grate. This ensures that only a relatively small amount of dust is entrained along in the hot air discharged from the cooler.
Along the entire length of the cooler, air is supplied from below through various undergrate compartments and flows through the grate plates and the bed of clinker, which is thus cooled. In order to attain low clinker exit temperatures, substantially more cooling air has to be blown into the cooler than is needed as combustion air by the kiln. The surplus hot air (cooling air heated on passing through the clinker bed) is utilized as exhaust air for other purposes or, after passing through a dust collecting unit with exhaust air fan, discharged into the atmosphere.
The fine clinker particles falling through the cooling grate, so-called riddlings, have to be extracted from the undergrate compartments through devices which seal off the escape of air from these pressurized compartments. Pneumatically actuated double-flap valves are more particularly used for the purpose. Alternatively, with low undergrate pressure, a drag-chain conveyor for direct removal of the riddlings may be installed in the bottom part of the cooler housing

3.3.1.2.2 Single-grate coolers

These are either of the inclined grate type (with slope of 3, 5 or 10 degrees) or the horizontal type. They can cope with up to about 1000 t of clinker per day The purely horizontal cooler is now virtually obsolete. Its drawback is that the rapid expansion of the cooling air in the hot part of the cooler tends to fluidize the fine-grained clinker on the horizontal reciprocating grate, so that the to-and-fro movements of the latter are rendered ineffective in moving the clinker along.
For throughputs above 1000 t/day the single-grate cooler is nowadays used only as pre-cooler operating with a gravity type after-cooler, or otherwise combination coolers are employed.

3.3.1.2.3 Combination coolers

The "combination" cooler comprises several (usually two) independent grates with their own drives and with separately controllable speeds. The hot part consists

of an inclined grate with a slope of 5 (or 3) degrees, while the after-cooling zone is provided with a horizontal grate. The inclined grate has a working width equal to about half the internal diameter of the kiln; it operates at a lower frequency and carries a clinker bed up to about 600 mm in depth. The horizontal grate is wider, having two or three rows of plates more, and is usually operated at a higher frequency of its reciprocating strokes, while the depth of bed on it is correspondingly less (about 250 mm).

3.3.1.2.4 Multiple-stage coolers with intermediate size reduction

This type of cooler is used mainly for clinker throughputs of 2500 t/day and upwards. It comprises and inclined grate, a short horizontal grate, an interposed clinker breaker, and a long horizontal grate.
The advantage of this form of construction is that, thanks to the intermediate size reduction by the air-cooled clinker breaker, intensive final cooling on the long horizontal grate is effected, enabling exit temperatures of 60°–80° C to be attained. Against this there is the disadvantage that this type of cooler occupies a greater amount of space and consumes about 0.8 kWh more electricity per tonne of clinker. In order to derive full benefit from the intermediate size reduction for exhaust air utilization, the cooler usually operates with a so-called duotherm circuit.

3.3.1.2.5 Air demand and duotherm circuit

Conventional clinker coolers are mostly operated with air ratings of between 2.1 and 2.8 Nm^3 per kg of clinker. Since modern heat-economizing kilns require less than 1 Nm^3 of combustion air per kg of clinker, these coolers discharge large quantities of exhaust air at relatively low temperatures and therefore unfavourable with regard to utilization of its heat content.
In the duotherm system, part of the hot exhaust air from the cooler is circulated back and introduced into the front air compartments. As a result of this recycling, the required intake of fresh cooling air can be reduced to 1.3–1.8 Nm^3/kg of clinker and the exhaust air rate can likewise be reduced, while its temperature is correspondingly higher, so that better economy in waste heat utilization is achieved. However, because of the intermediate dust collection required, the energy consumption, which for a conventional system is about 5.2 kWh/t of clinker (including the exhaust fan drive), is increased by about 0.8 kWh/t.
With the duotherm system in conjunction with an intermediate air-to-air cooler for the circulating air it is even possible to operate the grate cooler without any discharge of exhaust air at all.

3.3.1.2.6 Design dimensions; an example of an actual cooler

In view of the multitude of possible variants, the design of a reciprocating grate cooler will now be illustrated with the aid of an example allowing a comprehensive representation of all the more important features. See Fig. 15.

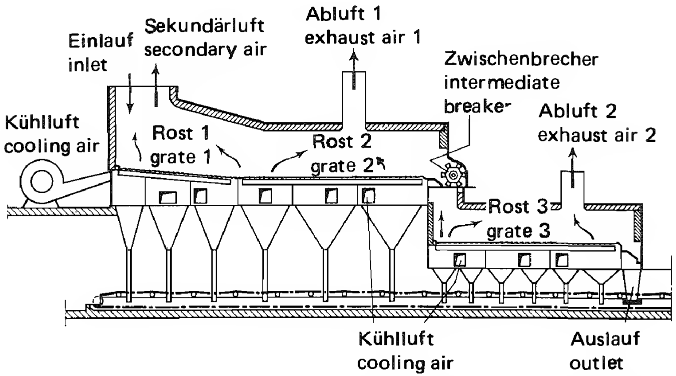


Fig. 15: Multiple-stage cooler with intermediate size reduction (from Steinbiß, 1972[1])

The example relates to a three-stage cooler, with intermediate size reduction, of the type built by the engineering firm of Claudius Peters. The principles it embodies are essentially applicable to all other reciprocating grate coolers, including for example:
the single-grate cooler (see Fig. 10);
the two-grate combination cooler, type Folax, built by FLS (Fig. 16);
the three-grate combination cooler built by Fuller (Fig. 17).

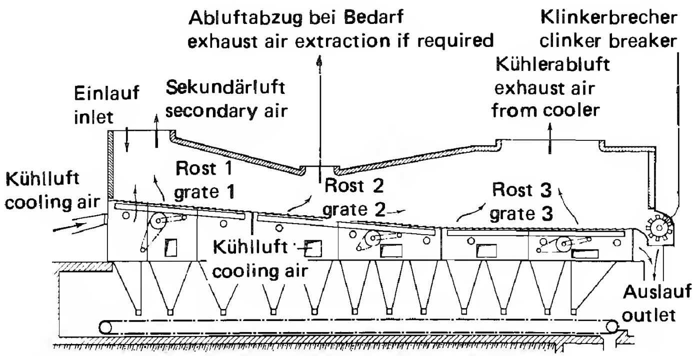


Fig. 16: Three grate combination cooler (from Erdmann, 1978)

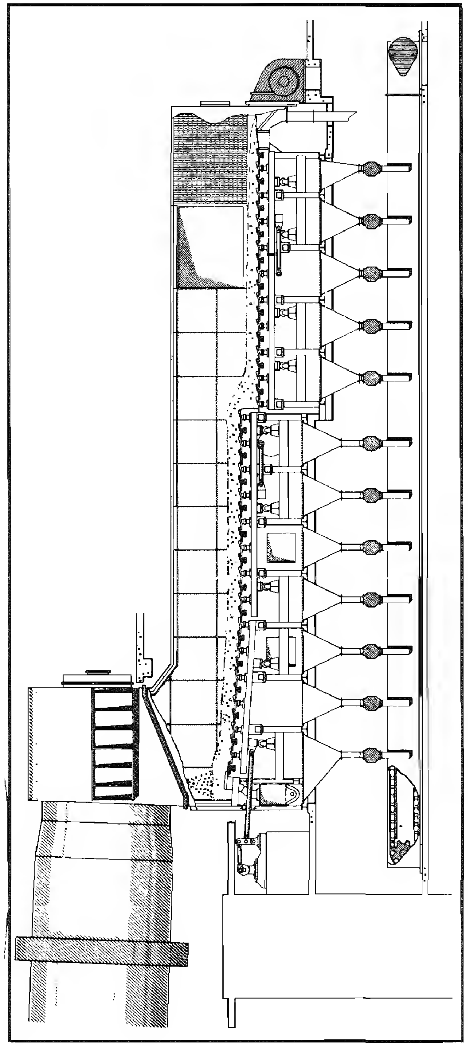


Fig. 17: Three-grate combination cooler (from Steinbiß, 1972[1])

A multiple-stage cooler with intermediate size reduction of the clinker is a complicated installation and economically justified only in connection with a heat-economizing kiln fed with meal and attaining a clinker output exceeding about 2000 t/day.
The three-stage cooler envisaged in the example is installed behind a preheater kiln and designed for dealing with 2100 t of clinker per day. Its designation is 825 S/1025 H/Zw/1033 H, a code denoting the main characteristics, as follows (see Fig. 18):

8	8 grate plates effective width	
25	25 ft length = 23 grate plates	grate 1
S	inclined (sloping) grate	
10	10 grate plates effective width	
25	25 ft length = 23 grate plates	grate 2
H	horizontal grate	
Zw	intermediate clinker breaker	
10	10 grate plates effective width	
33	33 ft length = 30 grate plates	grate 3
H	horizontal grate	

Cooling is accomplished in a residence time totalling about 40 minutes on the three grates, which have separate drives.
The narrow grate 1 forms the recuperation zone. The then following wider horizontal grate 2 is the precooling zone, to which preheated "duotherm" air is supplied. The clinker breaker installed behind this grate extends across the full width of the cooler After-cooling of the broken clinker is accomplished on the horizontal grate 3.
The cooler as a whole has a grate plate surface area of 78 m^2, so that the rating is 27 t of clinker per day and per m^2 The specific cooling air rate is 2.1 – 2.5 Nm^3/kg of clinker, while the intake of cold air from the surroundings of the cooler is only 1.7 Nm^3/kg of clinker. The average cooling air rating is 230 – 280 Nm^3/hour per plate or 2300 – 2800 Nm^3/hour per m^2.
The cooler is equipped with six fans for supplying the necessary air to the seven cooling compartments. With lowering temperature of the clinker bed along the length of the cooler the expansion of the air decreases and the specific rate of air per unit area of grate plate becomes correspondingly less. As a result of this the air pressure required for penetration of the bed likewise decreases. Each successive air compartment, from the inlet end of the cooler towards the outlet end, is therefore provided with a fan developing a lower static pressure than the preceding one. Thus, the first compartment has a fan developing 50 mbar (500 mm w.g.), while the fan for the last two compartments develops 15 mbar (150 mm w.g.).
The specific air supply rate is likewise graduated from the highest value in the first compartment (about 750 Nm^3/hour per plate) to 100 Nm^3/hour per plate in the last compartment.

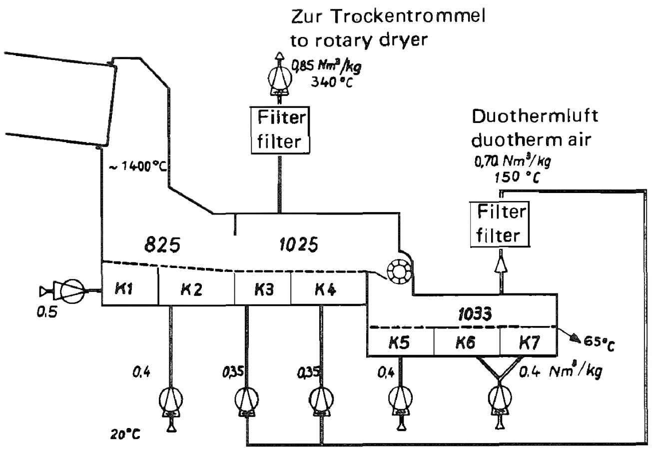


Fig. 18: Three-grate combination cooler with intermediate size-reduction

The respective pressures in the undergrate compartments, the air rates and the clinker temperatures are indicated in Fig. 19. More particularly in the front part of the cooler, where high air pressures are employed, the differences in pressure between successive compartments must not be too great, otherwise too much air blown into the first compartment, for example, will escape into the second compartment, for although the compartments are kept separate from one another, the air seals at the grate are never ideally effective in practice.

Recuperative zone

The recuperative zone of the cooler extends from the clinker entry point to the end of the first grate (in this three-stage cooler with intermediate size reduction). The other grates serve only for the final cooling of the clinker. In the inlet shaft to the cooler and on grate 1, which comprises about a quarter of the total cooling surface area, the clinker is cooled in about 20 minutes from about 1700° K to about 750° K. For this purpose two fans blow cold air – in a quantity corresponding to the total secondary and tertiary air required by the kiln – into the cooler, where this air is heated to about 1100° K, which is the temperature of the secondary air. Instead of cold air, preheated "duotherm" air can be used in order to achieve even better heat recovery for the kiln. However, the higher grate temperature rating that this involves will reduce the operational reliability of the cooler, so that in practice such an arrangement has proved too critical. For a heat-economizing kiln with good

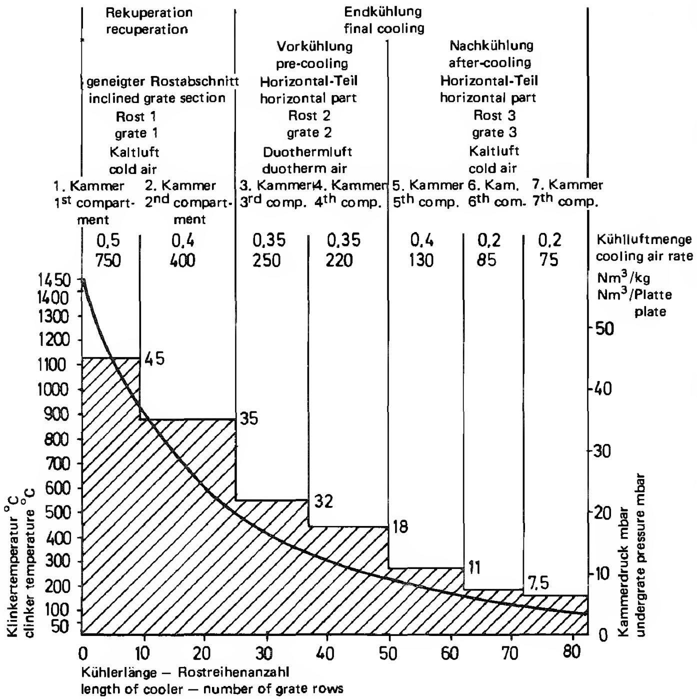


Fig. 19: Air and temperature conditions in a combination cooler

seals at the firing hood and low primary air rate (less than 10% referred to total combustion air) the secondary air rate, including any tertiary air required, is about 0.85 – 0.90 Nm^3/kg of clinker. The term tertiary air is applied to the air which is supplied direct to the firing system in the preheater of a kiln plant equipped for precalcining. This air should be extracted from a point close to the cooler shaft in the recuperative zone.

In order to achieve optimum heat recovery for the kiln system, it is necessary to prevent the air from the after-cooling zone from mixing with the secondary or the tertiary air. For this reason a partition to prevent transverse flow is provided at the end of the recuperative zone, in the hot air part of the cooler. Besides, from this point the roof of the cooler housing slopes upwards to the kiln and also in some systems upwards towards the clinker breaker, so as to obtain constant velocities

of the exhaust air and secondary air. The distance from the top of the clinker bed to the lower edge of the partition is about 1 m, so that there is sufficient headroom to allow occasional large lumps (fragments of collapsed clinker rings from the kiln) to pass underneath. From the viewpoint of durability it is advisable to construct the partition as an arched wall. See Fig. 20. On the downstream (cold) side of this partition an apron extending down to the top the bed may additionally be provided. This apron may consist of freely suspended plates of a heat-resisting steel such as Sicromal.

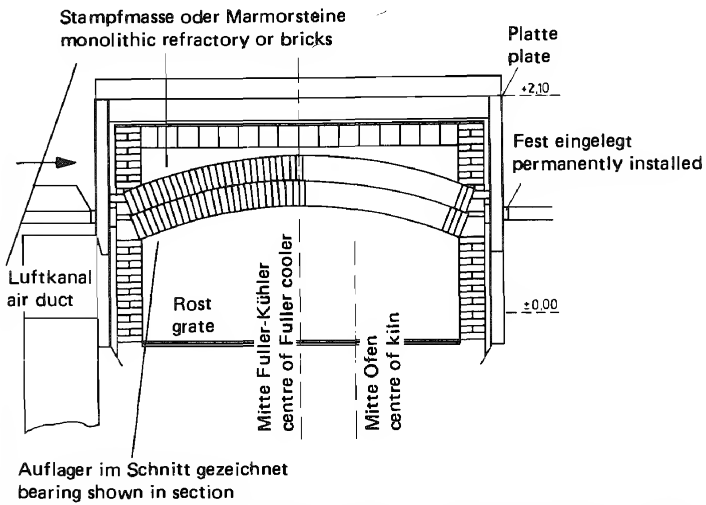


Fig. 20: Arched wall partition between recuperative zone and final cooling zone

Of primary importance for optimum heat exchange is to distribute the clinker as uniformly as possible across the width of the cooling grate. It has been found advantageous to align the cooler somewhat off-centre in relation to the kiln to allow for the non-central discharge of the clinker due to the kiln's rotation (Fig. 21).

The space over the cooler and inlet shaft should be amply dimensioned to keep the secondary air velocities below about 8 – 9 m/second, so that not too much dust is carried back into the kiln. The front part of the cooler itself should, on the other hand, be narrow, with a width equal to 0.5 – 0.6 times the effective diameter of the kiln, so that with a grate operating frequency of 6 – 10 strokes per minute a clinker bed of 500 – 600 mm depth is built up.

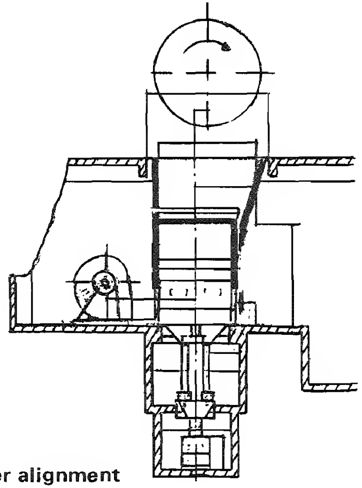


Fig. 21 : Kiln/cooler alignment

In practice it has proved advantageous to provide the front part of the cooler with an inclined grate. Its slope should preferably not exceed 3 degrees, however, so as to avoid any risk of uncontrolled rushing down of the clinker on it.
In the three-stage cooler with intermediate size reduction, as envisaged in the example, the housing is so dimensioned as to enable ten grate plates to be installed across the width. In the recuperative zone, i. e., grate 1, the width is reduced to eight plates by filling the space corresponding to one plate width on each side with monolithic refractory material to a height of about 500 mm.
The clinker discharged from the kiln should fall directly onto the clinker bed over the first few rows of plates. A deep bed is necessary not only for good heat recuperation, but also for protection of the grate itself against possible damage due to large pieces of coating detached from inside the kiln The discharge from the clinker is seldom very regular and, instead, varies with the thickness and shape of the coating formed at the kiln outlet, while more particularly in kilns fed with unpelletized meal there is a distinct segregating effect into coarser and finer clinker particles in the direction of kiln rotation Hence the grate plating in the zone which receives the clinker discharge from the kiln will have to be adequately suited to the operating conditions.
In earlier cooler designs a sloping inlet chute was provided, on which the clinker fell before rebounding into the actual cooler This chute was usually lined with a water-cooled distributing plate to prevent accretions of hot clinker ("snowmen"). Cooling this plate, however, caused a heat loss of about 0.063 GJ/t of clinker (15 kcal/kg).

In the three-stage cooler with intermediate size reduction under consideration there is a vertical front end wall. In order to prevent having locally too thin a bed of clinker, the first row of plates is stationary and the plates have no holes. The clinker that remains lying on these plates protects them and prevents the formation of localized air escape passages. Similar considerations apply to the edge plates of the first two rows not covered with monolithic refractory fill, which likewise have plates without holes. This is known as the "horseshoe" arrangement of plates. An example of this system in actual practice is shown in Fig. 22.

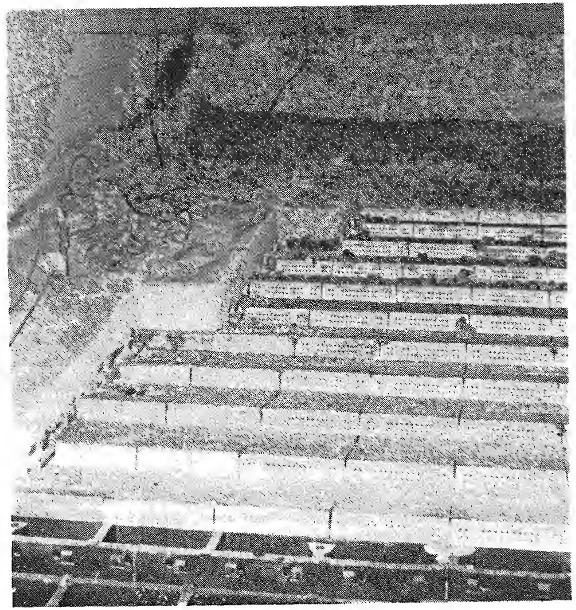


Fig. 22: Reduction in grate width

The effect of various plate arrangements on the air distribution and clinker bed is schematically illustrated in Fig. 23.
In contrast with the example discussed above, the recuperative zone is here, however, subdivided into three air compartments, the first of which has a length corresponding to only three rows of plates. With the high specific air rate supplied to this compartment the clinker on the grate is agitated in the manner of a fluidized bed, a condition which is claimed to achieve better distribution of the clinker.

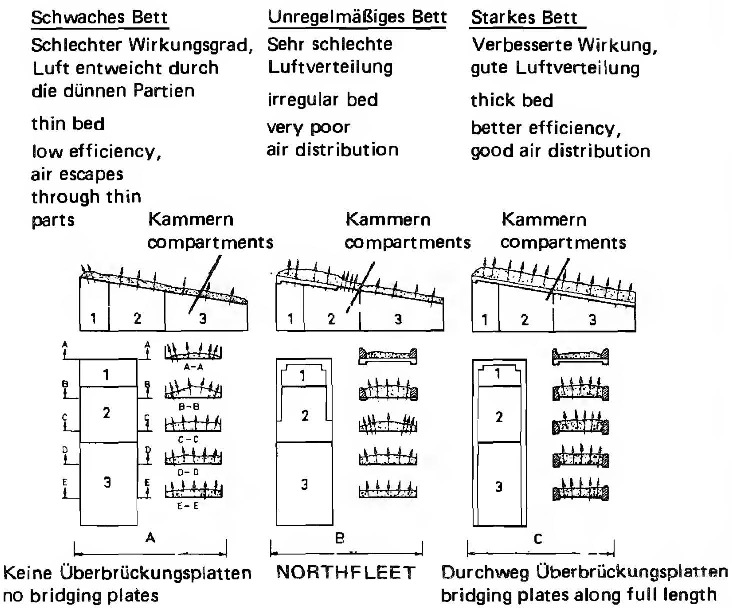


Fig. 23: Effect of plate arrangements on clinker bed and air distribution (from Ward/Watson, 1972)

Besides, the pressure in this compartment responds very rapidly to variations in clinker discharge from the kiln and can therefore suitably be utilized as a control variable.

In some instances the first compartment is divided longitudinally instead of transversely, with different air supply rates to the two longitudinal sub-compartments, an arrangement which is considered to counteract the tendency for the clinker to segregate into coarser and finer particles. In practice this has not proved to be a satisfactory solution, however, because it is not possible to form a sufficiently effective air seal between the two sub-compartments.

Besides the "horseshoe" plating system, other special arrangements for the plates have been devised for improving the air distribution and preventing the development of hot strands of clinker extending forward through the bed.

Among the more frequently adopted solutions are:

Bridging plates, with and without holes. With this system, certain plates are kept stationary in a movable row of plates, so that the clinker transporting action is arrested.

Curb plates: These are provided with an approximately 200 mm high raised edge curb and have a retarding and banking-up effect on the clinker movement. See Figs. 24 and 25.

These plates may be disposed either in checkerboard fashion on one side of the cooler behind one another, in order to retard or divert a continuous hot strand of clinker ("red river"), or they may, for example, be installed across the full width of the cooler in one or more rows in order to bank up the clinker bed. If curb plate rows are installed, it is necessary to ensure that the row with the highest curb is located at the end of a cooling air compartment, because the depth (thickness) of the clinker bed should be constant within each compartment. If the banking-up effect of the curb plates does not comprise the whole compartment, so that the bed in the rear part is thinner than in front, the air flow will be concentrated in this rear part, while the thicker bed will get correspondingly less of the air and the clinker will be less effectively cooled. To prevent local air breakthroughs it is also important that a row of curb plates is always followed by a row of plates without holes. Nor should there be any holes in the plates located in the corners directly behind the portions of the grate which are filled in with monolithic refractory.

In the example of the three-stage cooler with intermediate size reduction, the last row of plates in grate 1 and the first row in grate 2 are stationary rows. In this way a banking-up effect is obtained, so that the clinker does not rush too quickly onto the

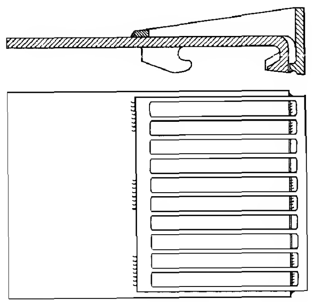


Fig. 24: Plate with welded-on raised edge curb (from Steinbiß, 1972[1])

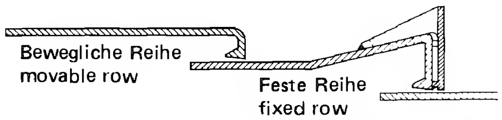


Fig. 25: Wearing or curb plates on fixed grate plates (from Steinbiß, 1972[1])

two extra outermost longitudinal plate rows of grate 2 (which has an effective width of 10 plates, as compared with only 8 for grate 1).
In order to improve operational reliability it is advantageous to install a valve between the cooling fan and the cooling air compartments and also to provide closable doors between the respective compartments. With these precautions, one fan can be stopped in an emergency without having to shut down the kiln.
Because of the high air pressures in the recuperative zone, it is not possible, in this part of the cooler, simply to use a drag chain conveyor for removing the grate riddlings (the fine particles that fall through the slots in the grate); very efficiently closing clinker discharge locks are required instead. Pneumatically operated double-flap valves have been found satisfactory for the purpose. Electric motors are not suitable for operating them because of their limited working life in the dust-laden atmosphere.
The valves under the individual collecting hoppers open either in a predetermined sequence at certain intervals or in response to the level of the material in the hoppers, which operates a control system. In the latter case, however, the valves must be interlocked so as to make sure that several valves will not open simultaneously and overload the clinker handling equipment.

Final cooling zone

The recuperative zone is followed by the final cooling zone, where the clinker is cooled from about 750° K to its final temperature. The grate area for final cooling is nearly three times as large as that for heat recuperation. The reason for this is the cooling rate, which decreases as the difference in temperature between the cooling air and the clinker to be cooled becomes less.

Precooling grate

The horizontal grate of the precooling zone is subdivided into two compartments and is supplied with "duotherm" air with a temperature of about 150° C. The heated air is extracted over grate 3, which forms the after-cooling zone behind the intermediate clinker breaker, and passed through a mechanical dust collector for protecting the cooling air fans against excessive dust load. With the duotherm system of air control the exhaust air is thermally upgraded, i. e., the volume of air is reduced and its temperature raised, enabling its heat content to be more effectively utilized for material drying. The exhaust air should be extracted, not from one side of the cooler, but preferably through the roof, so as to carry along as little dust as possible. The cooling air rates are so adjusted that grate 2 gets less air than grate 3 and that part of the exhaust air is supplied in the form of air overflowing from grate 3. This overflow of air serves also to cool the clinker breaker.
However, as a result of the admission of the hot duotherm air the cooling curve in the precooling zone presents a flatter shape than in the recuperative zone and in the after-cooling zone.

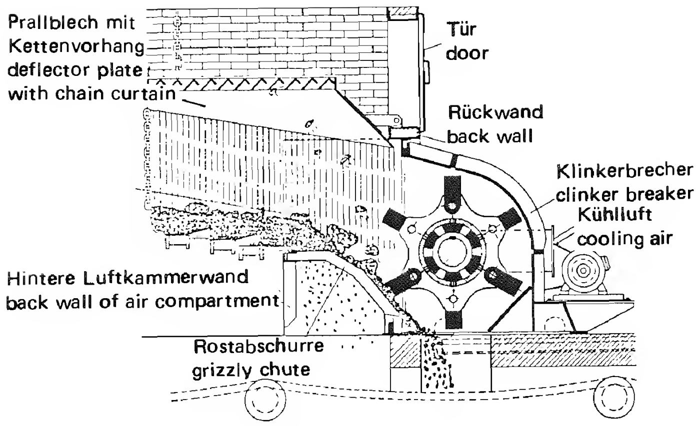


Fig. 26: Intermediate clinker breaker (Cl. Peters GmbH)

Intermediate clinker breaker

This is a single-rotor hammer crusher (Fig. 26) with an air-cooled hollow shaft, the air being supplied by an external fan. Alternatively, so-called autogenous cooling may be used. In this system the plates which are mounted on the shaft and carry the hammers are so interconnected as to form a cooling air duct extending across the full width of the grate. The duct is provided with numerous openings through which the air it draws into its interior can flow out.

After-cooling zone

On passing through the breaker, the clinker is substantially increased in surface area and discharged onto grate 3, on which it forms a bed of completely rearranged particles and where, with a relatively moderate quantity of cooling air, it is cooled from about 550° K to its final temperature of below 350° K. In order to minimize energy consumption, this after-cooling grate is operated with a low bed depth and correspondingly low air pressure. For this reason, too, there is no need for double-flap valves to act as air locks to the compartments in this part of the system. A drag chain conveyor for removing the grate riddlings can pass direct through the compartments.

Heat balance and heat flow

The heat flow and balance for the clinker cooler considered in the example are indicated in Figs. 27 and 28.

	with intermediate breaker (kcal/kg)	GJ/to
heat supplied	(358)	1,50
heat expenditure		
clinker	(8)	0.03
radiation and convection	(6)	0.03
exhaust air	(86)	0.36
secondary air	(258)	1.08
efficiency		
internal	72	
external	96	

Fig. 27: Combination cooler with intermediate size reduction: heat balance

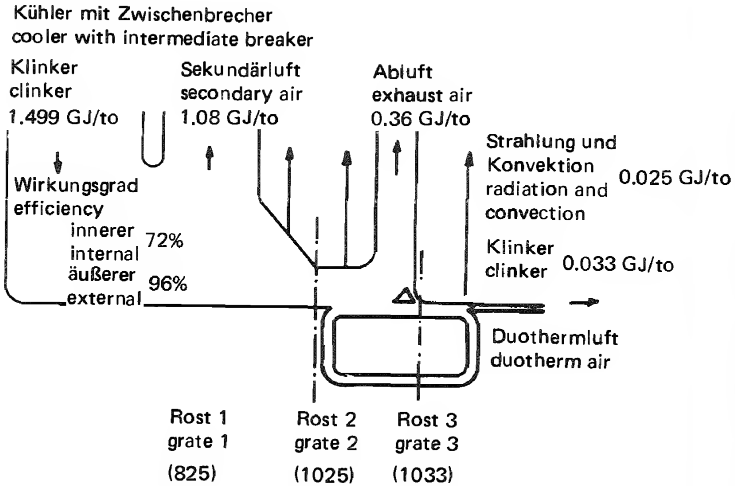


Fig. 28: Cooler with intermediate breaker: heat flow

Cooling air and exhaust air fans

Cooling air fans

The correct choice of cooling fan capacity is of major importance with regard to the efficiency and the electric energy or power consumption of the cooler.

Good heat recovery is possible only if the cooler is operated with a deep bed of clinker, with a correspondingly high air flow resistance. For this reason the fans in the hot zone should be able to develop a sufficiently high pressure. The required minimum pressures for the fans supplying air to the recuperative zone are as follows:

for single-stage coolers:	30 mbar
for multiple-stage coolers:	50 mbar.

The other cooling air fans should be of correspondingly lower performance in terms of pressure developed, reduced stepwise to 10 mbar, as exemplified in Fig. 19.

The electric power consumption of the cooling air fans accounts for more than 70% of the overall power demand of the cooler. Hence it is essential to use high-efficiency fans which are properly suited to the actual operating conditions of the cooler.

Single-inlet radial flow fans are most suitable for the purpose.

At the design operating point the fan efficiency should be between 70 and 80%. Curved-blade impellers are most appropriate. Protection against wear is important only for fans which have to handle duotherm air or air for the kiln hood seal.

As the air pressure to be developed by the fans varies greatly with the specific load of the cooler, fans having not too flat a characteristic curve (pressure-volume curve) should be chosen. Simple damper control is uneconomical; it is preferable to use inlet vane control.

An inlet vane control system (Fig. 29) comprises a static guide-vane unit which is installed in front of the fan's impeller and whose radial vanes can be swivelled by means of a control device so as to vary the inlet air flow conditions. These vanes deflect the inflowing air in the direction of rotation or in the opposite direction. As a result of this preliminary guidance, the entry losses are substantially less than those associated with ordinary damper control.

The difference in power consumption between throttling down the inflow by means of a damper and inlet flow control with guide vanes is apparent from Fig. 30.

The conditions for fans with inlet vane control which are, respectively, well adapted and unfavourably adapted to the operating requirements are indicated in Figs. 31 and 32. In these diagrams the static pressure of the fan has been plotted against the volume flow for various guide-vane settings. The dot-dash lines indicate the efficiency of the fan.

In case 1 the operating point is at a pressure of 42 mbar and a flow rate of about 46 000 m^3/hour. The vane setting is 45 degrees and the efficiency is over 70%. The pressure developed by the fan corresponds approximately to the air pressure in the cooling air compartment. It is evident that in normal service the fan is working at a favourable operating point and that nevertheless adequate reserves are available to cope with increases in clinker bed resistance due to variations in clinker particle size or discharge rate.

Different conditions exist in case 2. Here the operating point already in normal service is at a guide vane setting of 0 degrees. The efficiency is unfavourably low,

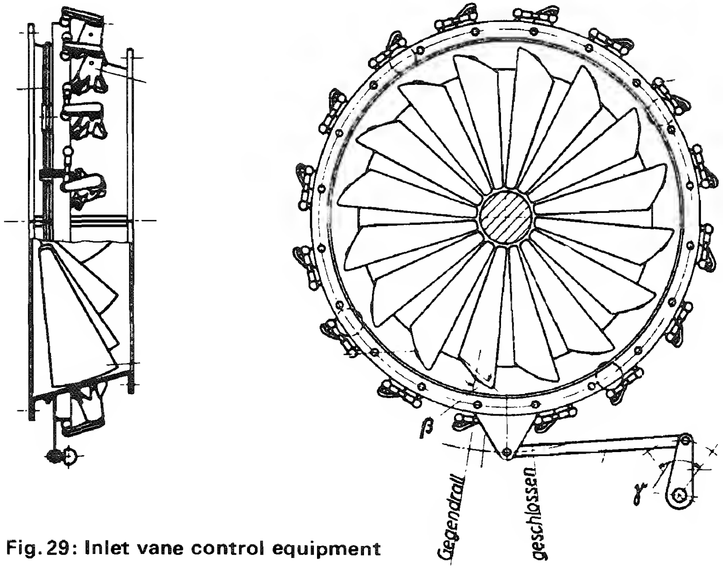


Fig. 29: Inlet vane control equipment

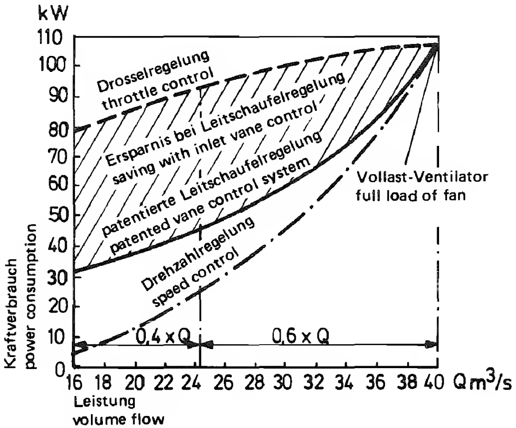


Fig. 30: Power consumption

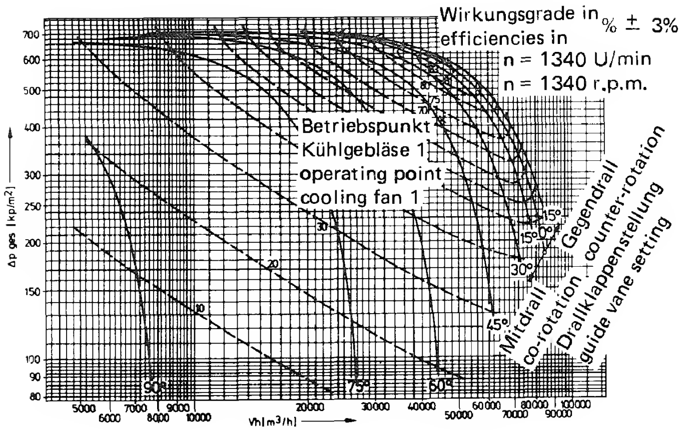


Fig. 31: Favourable adaption of fan to requirements

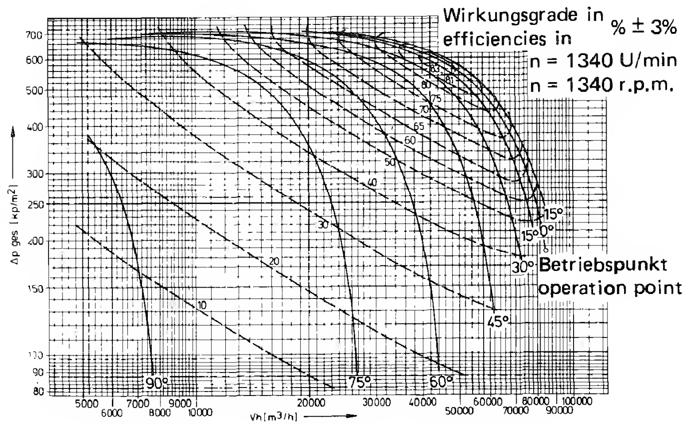


Fig. 32: Unfavourable adaption of fan to requirements

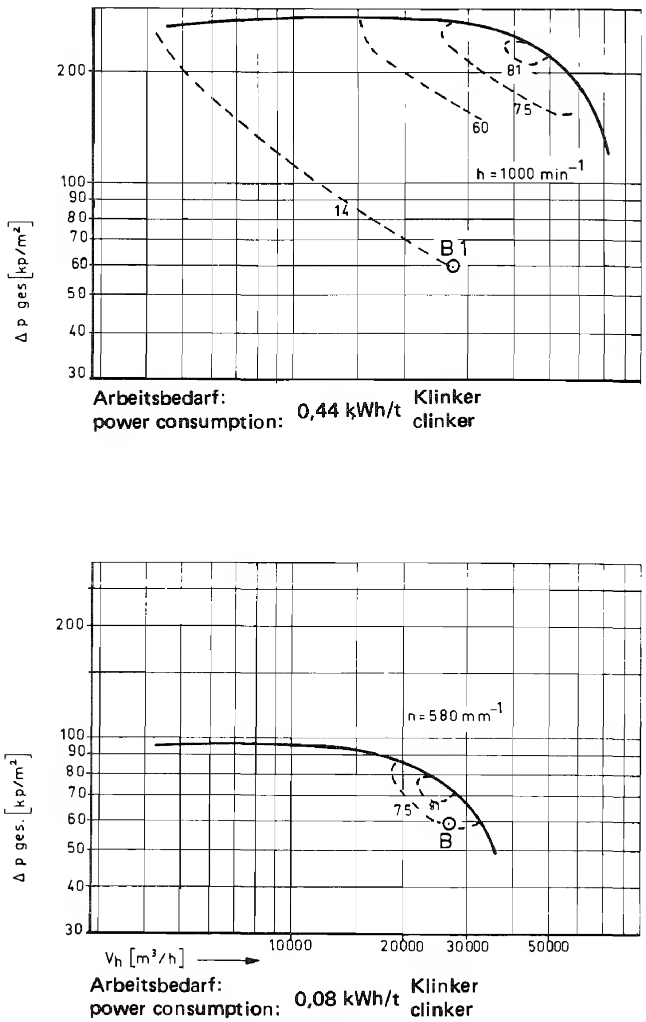


Fig. 33: Adjustment of the characteristic by speed changing

only 40%; even so, no pressure reserves are available to cope with off-normal conditions. In the event of a rise in pressure in the associated cooling air compartment, e. g., due to an increase in the rate of clinker discharge from the kiln, the cooling air supply rate will therefore decrease, with the attendant danger that the cooler will be overheated.
In many actual instances the future operating conditions cannot be predicted with sufficient accuracy at the design stage. It is therefore advantageous to provide the fan with a vee-belt drive, enabling it to be adapted quite simply to the actual operating requirements by changing the vee-belt pulleys and thus altering the speed. This is illustrated in Fig. 33.
A further saving in power consumption can be achieved by the use of an inlet nozzle, which reduces the entry losses, besides providing a convenient means of measuring the rate of flow delivered by the fan. See Fig. 34.

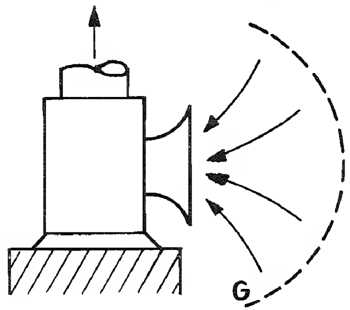


Fig. 34: Inlet nozzle

Exhaust air fans

For the sake of operational reliability this fan should be of very ample capacity, as the operating conditions are subject to frequent and rapid variations. When coating becomes dislodged in the kiln, for example, the temperatures of the air to be handled may rise by more than 200° K, with a corresponding reduction in the fan's air delivery rate.
In order to cut down electric power consumption it is therefore nearly always economically advantageous to use an adjustable-speed motor for driving the exhaust air fan.
Besides, a reliable water injection system is to be recommended, in order to protect the fan, the dust collection system (on the downstream side of the fan) and the clinker handling equipment against overheating in extreme cases.

Water injection into the cooler

Except in the manufacture of white cement, water is used only for after-cooling the clinker. The recuperation of heat must not be affected by it. The cooler should be so amply designed that it will be necessary to have recourse to water cooling only

under extreme conditions or in the event of a fault, e.g., to cope with excessive clinker discharge from the kiln as a result of coating collapse. Under such circumstances the injection of water will serve to protect the dust collection equipment and/or the handling appliances. Very small amounts of finely sprayed water may also be allowed in normal continuous operation of the cooler. Indeed, if the exhaust air is dedusted in an electrostatic precipitator, such water is necessary for conditioning the air.
Water injection equipment must be carefully designed and properly serviced. Too much or poorly atomized water sprayed into the cooler is liable to cause serious trouble because the very fine particles of clinker will react with the water. As a result, for example, the air holes in the grate plates may become choked or the hoppers and discharge devices for the grate riddlings may become blocked solid with hardened masses, while clogging of fabric filters may also occur.
The following are some important design criteria for water injection systems.
Arrangement of the nozzles. The nozzles should be located on the roof or on the outlet end wall of the cooler. There should be several nozzles, whose jets should not overlap. Wetting the clinker must not begin earlier than at least five rows of plates past the recuperative zone and must be completed at least five rows ahead of the discharge end of the grate, so as to ensure that recuperation will not be impaired and that all the water will have evaporated before the clinker leaves the cooler.
Type of nozzle: Nozzles should be either of the pressure-jet atomizing type or the twin-fluid type with compressed air atomizing. In the latter case the nozzles are supplied with water and atomizing air. The jet should be fan-shaped and very fine. It must not be allowed to impinge on the walls of the cooler
The nozzles and control valve should be so designed that adequate atomization is achieved even when operating at the lowest water feed rate. When the water jet is turned off, the nozzle should be blown clear with compressed air, and while the cooler is operating without water injection a scavenging air blower should ensure that the nozzles are at all times kept free from clogging with dust. An automatic device, actuated by a pneumatic cylinder, for retracting the nozzles when not in use has also proved advantageous. Water injection should be started automatically through the agency of a control system in response to the exhaust air temperature.
Nozzles, control valves, water pump and compressed air supply system (if any) should be of such capacity that, in the event of a temporary increase of up to about 50% in the clinker discharge rate from the kiln (e.g., surge conditions due to dislodgment of coating), the exit temperature of the clinker at the outlet of the cooler and the exhaust air temperature do not rise by more than 100° K in relation to normal operation The water feed rate for ensuring this is about 0.15 kg per kg of clinker at rated throughput. If the dust collection equipment for the exhaust air is a fabric filter, a 100° K rise in air temperature can generally not be tolerated In that case the air-to-air cooler installed upstream of this filter should be of such capacity that the exhaust air temperature rise which occurs despite water injection into the clinker cooler can be cancelled here. If no such air cooler is provided, the exhaust air fan should be of such capacity as to achieve the necessary cooling of the exhaust air by the addition of cold air from outside the clinker cooling system.

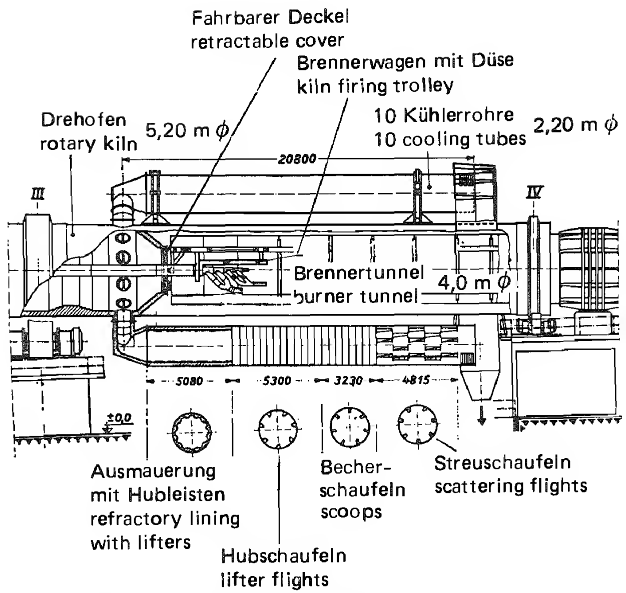


Fig. 35: Planetary cooler (from Kadel, 1974)

3.3.2 Planetary coolers

The planetary or satellite cooler, a long-established type of clinker cooler, has in recent years re-emerged in improved high-capacity versions which have secured a substantial share of the market in conjunction with new heat-economizing plants and is now available from nearly all major cement machinery manufacturers.
It is characterized more particularly by its, in principle, simple form of construction. It has no cooling air fans and no separate drive, being rotated with the kiln. See Fig. 35.
A planetary cooler consists of a number of cooling tubes, usually ten, disposed around the circumference of the kiln shell. Each of these tubes is connected to the kiln via a special elbow-shaped inlet through which the clinker passes. At the outlet end of the planetary cooler its tubes (in the newer designs of such coolers) are supported on the kiln shell, which is extended for this purpose and provided with an additional roller stand to carry the extra weight.
Access to the outlet of the kiln is obtained through a tunnel formed by a stationary tube projecting into the kiln shell extension. This tunnel is thermally insulated

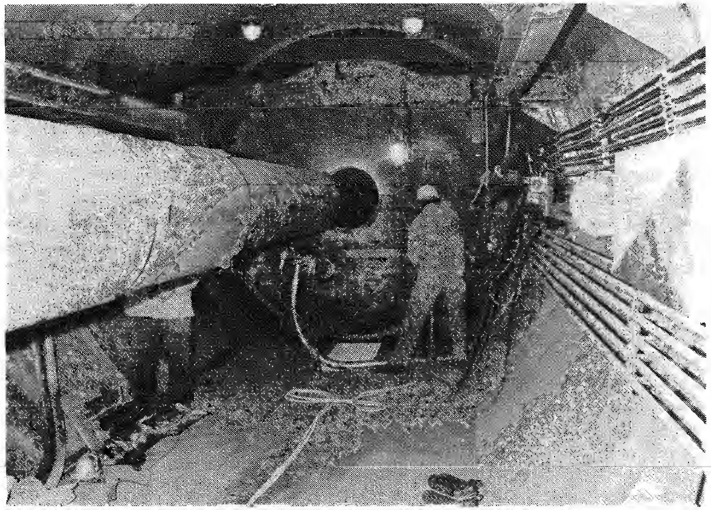


Fig. 36: Planetary cooler, tunnel

against heat penetration into it from the shell, and the tunnel floor is of hollow construction and accommodates air ducts for cooling it. With these arrangements it is possible to approach the outlet end of the kiln and the firing pipe installed there. See Fig. 36.

The hot end of the kiln is closed by a refractory-lined cover which is mounted on runner wheels and rails in the tunnel and is normally kept pressed against a seal on the kiln by the action of counterweights or pneumatic cylinders.

Each tube of the planetary cooler is mounted on two supports attached to the kiln shell. The inlet elbows, through which the hot clinker is discharged from the kiln into the cooling tubes, and the front part of the tubes themselves are lined with refractory material embedded in which are lifting ridges (made of ceramic refractory) and steel breaking teeth and lifters ("flights" or "scoops"). The refractory-lined zone of each tube is followed by unlined zones equipped with further scoops and lifting devices whose material and shape are suited to the various service conditions encountered.

The outlet ends of the cooling tubes rotate in the discharge end housing, where the clinker passes along a screening grid incorporated in the end of each tube. The coarser clinker lumps retained on the grid is fed to a crusher.

In process engineering terms the planetary cooler functions on the same principle as the rotary cooler. The cooling air rate corresponds to the secondary air supplied to the kiln. The air flow through the cooler is sustained by the kiln exit gas fan.

However, as contrasted with the rotary cooler, in the planetary cooler the movement of the clinker is governed by the rotation of the kiln. There is the further difference that the flow of clinker is divided among the respective cooling tubes. As in the rotary cooler, the air velocity is allowed to be varied only within a fairly narrow range, so as to achieve good heat transfer and to prevent cyclic movement or congestion of fine clinker particles.

The following approximate dimensional relationships are widely adopted in planetary cooler design:

throughput: 3–4 t of clinker per day and per m^3 of the total volume of the cooling tubes;

length/diameter ratio: 9:1 to 11:1.

The diameter of the individual cooling tubes is chosen in relation to the kiln diameter. For example, the planetary cooler on a kiln 3.8–4.4 m diameter will have tubes of 1.5–1.7 m diameter. On a kiln of 5–6 m diameter they will be in the range of 2.0–2.5 m.

The planetary cooler has no independent drive, the power to rotate it being provided by the kiln drive, which therefore has a correspondingly higher power consumption as compared with a kiln equipped with a grate cooler or rotary cooler. Also, the kiln exit gas fan, which has to sustain the air flow through the cooler, will have a higher power rating. In general, the additional energy requirements for planetary coolers fitted to heat-economizing dry-process kilns are approximately:

0.7–1.5 kWh/t of clinker for the kiln;

0.3–0.5 kWh/t of clinker for the exit gas fan.

On account of the high mechanical and thermal loads involved, the structural design of the planetary cooler is especially important. The following are more particularly regarded as problem zones:

- the kiln shell in the vicinity of the clinker discharge ports, i.e., the openings through which the clinker enters the cooling tubes;
- these openings with their cast steel inlet sockets, seals and elbows;
- the arrangements for attaching the cooling tubes to the kiln shell;
- the steel shells of the tubes with their internal fittings.

Kiln end section with inlet sockets

The openings (discharge ports) in the kiln shell at the planetary cooling tube inlets mechanically weaken it. This must be compensated by local increase in shell plate thickness (up to 60 mm in kilns of 3.8–4.6 m diameter, at least 80 and ranging up to 100 mm for kilns of 4.6–5.6 m diameter). See Fig. 37.

Furthermore, in order not to reduce the width of shell plate between the openings too much, the latter are oval in shape and are protected by inserted sockets made of heat-resisting cast steel. In a sense, they correspond to the outlet sectors (nose sectors) of rotary kilns with other types of clinker cooler and are embedded in refractory lining material. The refractories that have achieved the best results in this

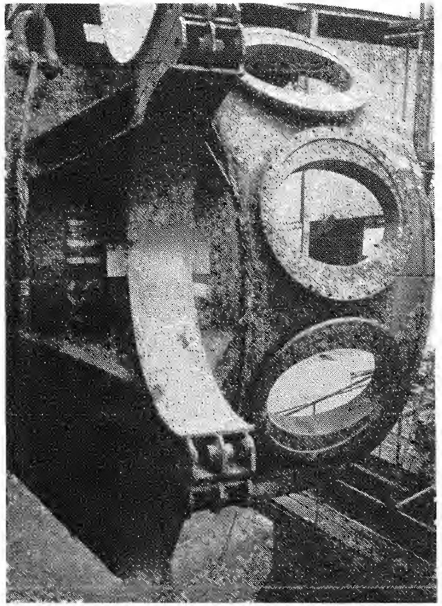
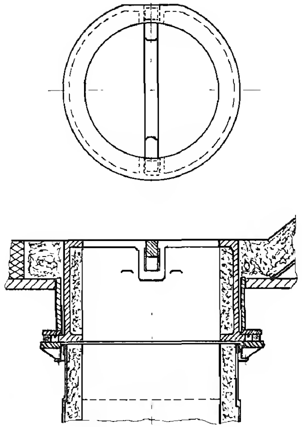


Fig. 37: Kiln end section

part of the kiln are high-alumina/corundum monolithic castable materials. They do, however, require very slow and careful drying and heating.
It has been found advantageous to provide loose breaker bars (Fig. 38) in the inlet sockets, achieving better wear behaviour and preventing excessively large lumps of clinker from entering the cooler. Such lumps, more particularly arising from dislodged coating, could choke the inlet elbows and damage the internal fittings (lifters) in the cooler. In order to prevent them from having a lifting action, these bars are placed perpendicularly to the longitudinal axis of the kiln.
Abnormally high temperatures in the discharge zone of a planetary cooler kiln are especially critical. For this reason it is essential to apply continuous temperature monitoring for the detection of adverse thermal conditions. Failure to do this is liable to result in damage to the refractory brickwork, involving very expensive repairs. Cracks may form in the kiln shell itself, and the cooling tubes are subjected to unequal operating conditions. Some of them will moreover be thermally overloaded, with reduced working life of the refractory lining, steel liner plates and internal fittings.
A refractory dam ring, built of brick, in the kiln shell has been found helpful in protecting the discharge zone (Fig. 39). A properly located and constructed dam

Fig. 38: Inlet socket with breaker bar



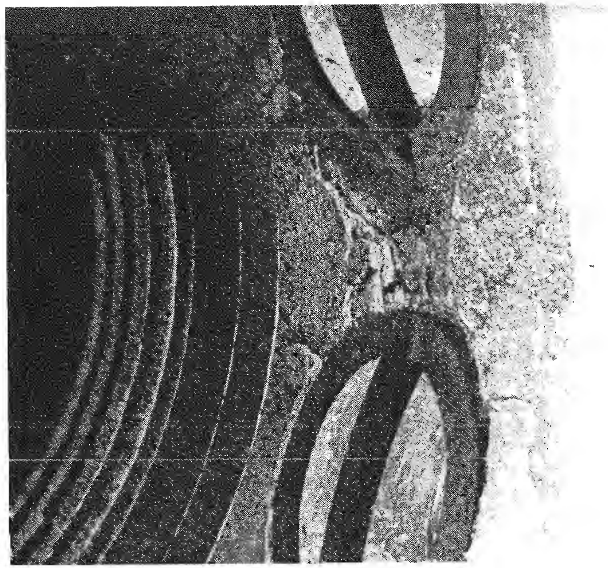


Fig. 39: Refractory dam, built of brick

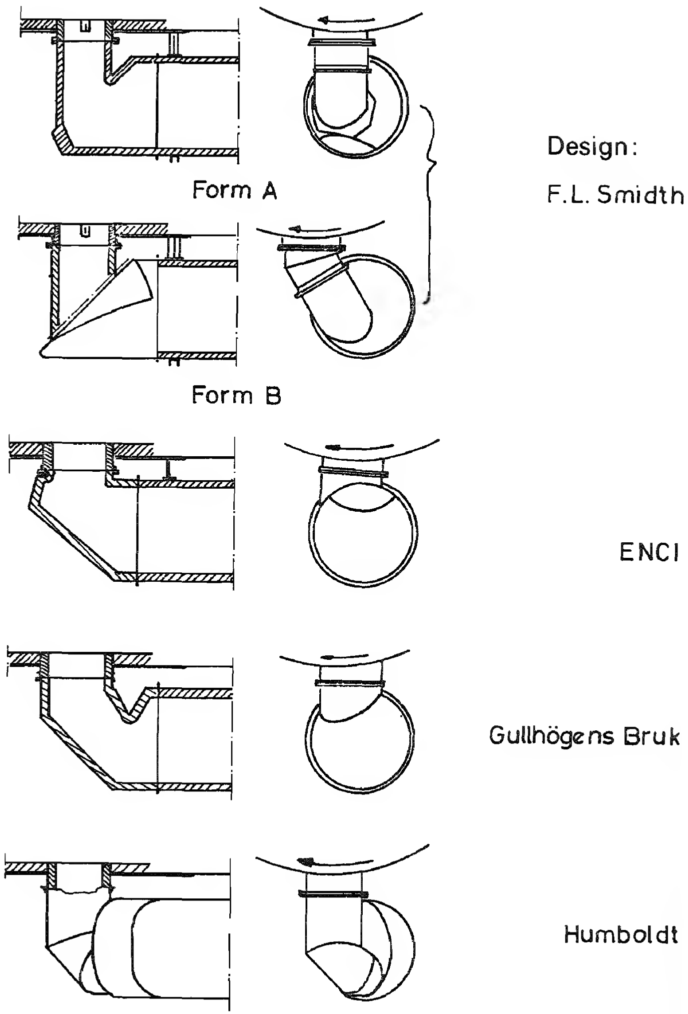


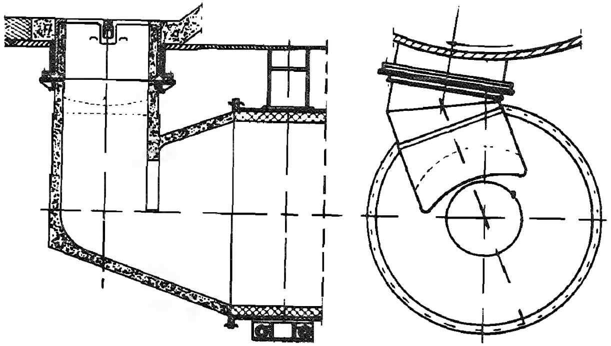
Fig. 40: Various forms of construction for inlet elbows

ring offers the following advantages:
protection of the kiln shell section carrying the tyre at the outlet end against overheating;
increased retention time of the clinker in the kiln, so that it enters the cooler at a lower temperature;
more uniform discharge of the clinker into the cooling tubes.

Inlet elbows

These are the ducts through which the clinker passes from the kiln into the tubes of the planetary cooler. They have to meet a number of requirements:

The clinker should be discharged as quickly as possible into the tubes and not fall back into the kiln during the course of each revolution. The height of fall of the clinker should be low, clinker should fall on clinker, if possible, and abrupt changes of direction should be avoided, so that wear by the abrasive action of the dust-laden secondary air is kept to a minimum.
Finally, the parts should be easily demountable and convenient to exchange for replacement parts without necessitating any modification.
Fig. 40 shows various designs for the inlet elbows to the cooling tubes. The form of construction in which the elbow is placed somewhat off-centre in the direction of rotation, and that in which a bridge or weir is provided as an internal fitting in the elbow, have proved most suitable (Figs. 41 a and 41 b).
The inner walls of the inlet elbows are subject to conditions of severe abrasive wear, and for this reason a good durable refractory lining is especially important. If the shape of these parts allows it, a lining of mullite brick is likely to give the best performance in terms of trouble-free operation and durability. Monolithic refractories have also been used with success, more particularly: wear-resistant high-alumina/corundum castables or rammed monolithic refractories developing chemical and ceramic bond. Whereas the construction of linings with castable refractories in suitable formwork is a fairly quick operation, the use of ramming mixes is slow and requires skilled manpower. Layer-by-layer ramming is moreover liable to result in spalling-off of flat pieces of refractory in subsequent service of the cooler.
If chemical/ceramic bonding monolithic refractory is used, it is moreover essential to preheat the freshly lined inlet elbows under controlled conditions, keeping a close watch on the temperature and maintaining a temperature gradient of not more than about 25° C/hour. This is because the chemical phosphate bond develops only at temperatures above 200° C, and it takes an even much higher temperature (1000° C and above) for the ceramic bond to develop. If the elbows are mounted directly after being lined, i. e., without preheating, the refractory will not be sufficiently heated by the kiln burner, and the necessary bond temperatures will therefore not be attained until hot clinker is admitted. By that time, however, damage may occur because the lining will not yet have gained adequate wear resistance.



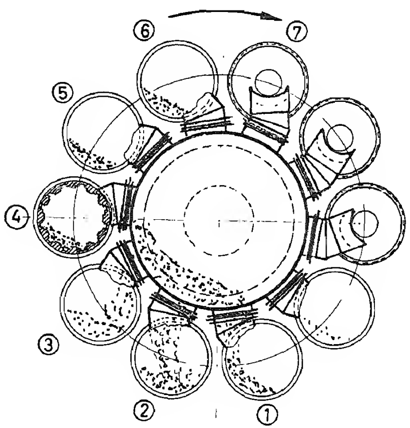


Fig. 41 : Inlet elbows

Elbow-to-kiln joint

The inlet elbow has to be connected in a positive and restraint-free manner to the kiln. Variations in length both parallel and perpendicular to the kiln have to be compensated, and the forces due to heating and deformation must not be transmitted across the joint.
The seal at the joint should be effective so that clinker dust cannot escape. Fig. 42 shows some commonly employed forms of joint.

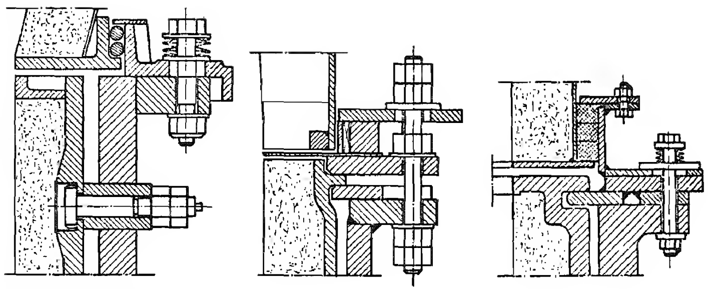


Fig. 42: Elbow-to-kiln joint

Cooling tube mountings

The tubes of planetary coolers on modern rotary kilns are always provided with two supports or bearings per tube. At one end is the fixed bearing, locating the tube axially and also preventing its rotation about its own axis. The movable bearing is able to accommodate the changes in length of tube due to temperature differences. Particularly at this bearing it is essential to have a sufficiently strong mounting construction, and the kiln shell plate should be sufficiently rigid (not less than 60 mm thick), as should also be the cooling tube (plate thickness not less than 20 mm). Attempts to relieve the movable bearing by applying additional lubrication to ease its working conditions have not been successful.
Three solutions for the fixed bearing are shown in Fig. 43. Forces due to longitudinal thrust (caused by the slope of the kiln) and to deflection of the cooling tube occur during the course of each revolution of the kiln. In the first and third solutions these forces are resisted by robustly dimensioned wide bearing stools. In the second solution there are likewise wide stools, but in addition the suspension stirrup enclosing the cooling tube is designed to tilt within certain limits, so that there is some "give" in the tube during each revolution and the shear forces are thus reduced.

Internal fittings in the cooler

With regard to the appropriate choice of internal fittings in the tubes of a planetary cooler it is important that the kiln itself should be equipped with a firing nozzle that extends 6 – 10 m into the kiln, so that there is a substantial length of kiln which can serve as a preliminary cooling zone and that the temperature of the clinker on entering the cooler itself is correspondingly lower. This will reduce the severity of the thermal conditions in the cooling tubes.
Typical temperature curves for the clinker, secondary air and kiln shell are shown in Fig. 45.

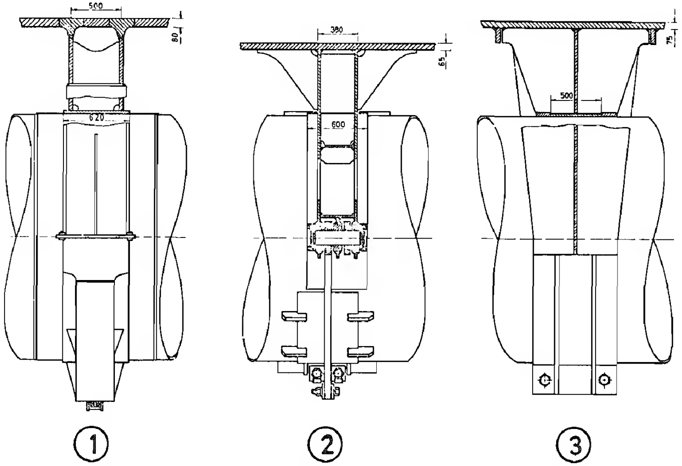


Fig. 43: Cooling tube mounting: fixed bearing (from Münk)

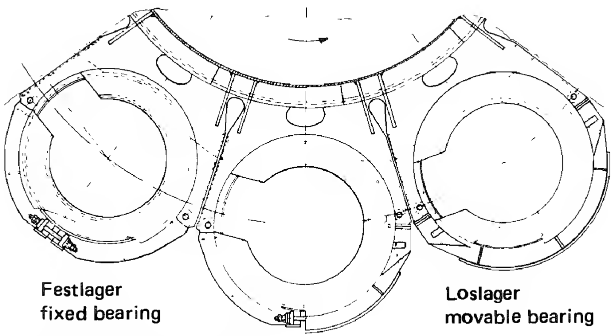


Fig. 44: Cooling tube mounting

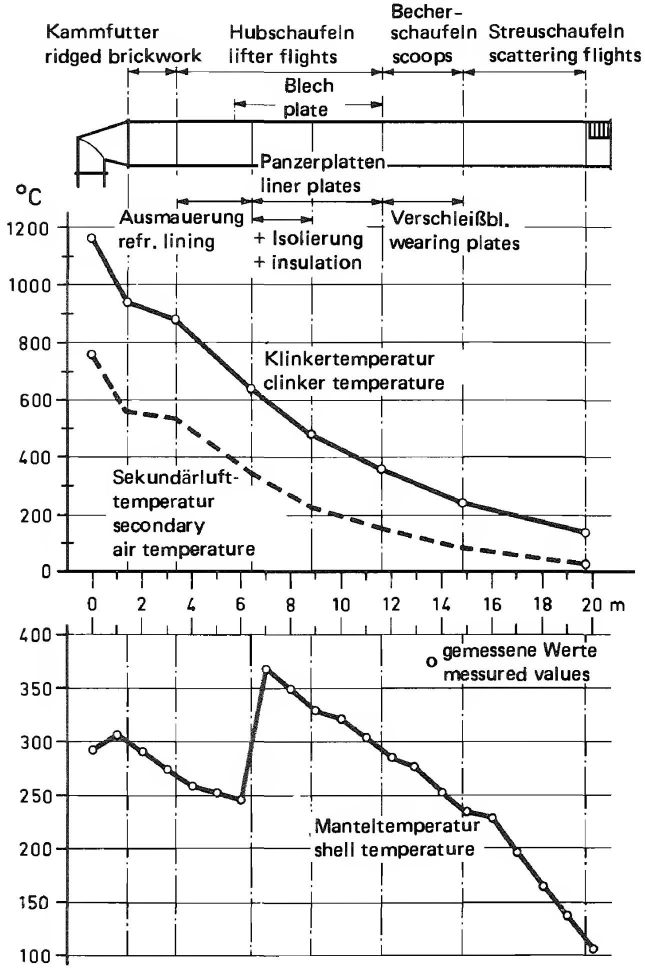


Fig. 45: Typical temperature curves for the clinker, secondary air and kiln shell

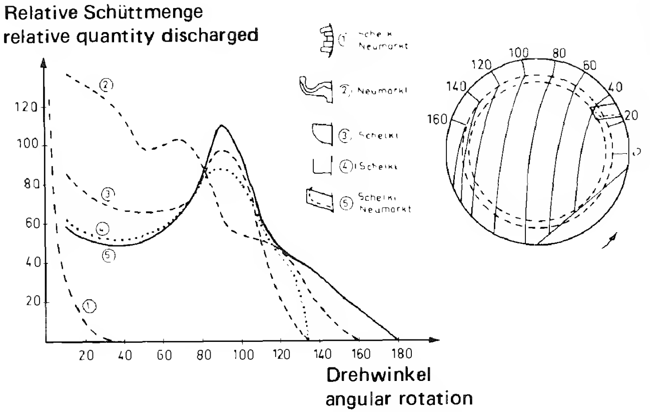


Fig. 46: The action of some commonly used internal fittings

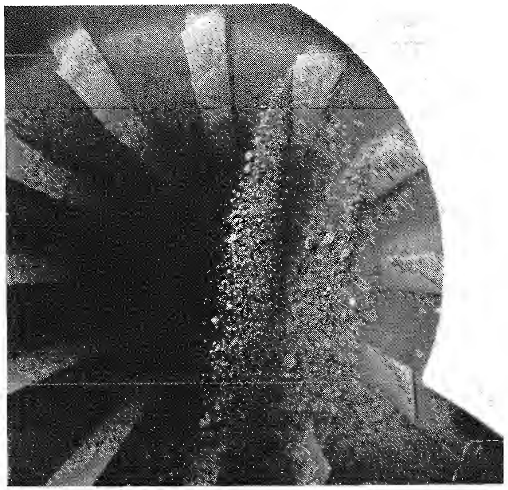


Fig. 47: Lifter flights are emptied too soon

It should be borne in mind that substantially higher temperatures may temporarily occur if the clinker is unequally distributed among the cooling tubes or if – in consequence of surge produced by dislodged masses of coating from within the kiln, for example – the total rate of clinker discharge suddenly increases. For this reason, too, the steel plate thickness of the tubes should not be less than 15 mm. In addition, as an extra safety precaution, the critical zone directly behind the refractory-lined part of the tube should be fabricated from a better grade of steel (e.g., 15 MO 3).

Another factor to be considered is that, in contrast with the transverse-flow cooling effected in grate coolers characterized by good heat recovery and relatively low rates of wear, planetary coolers operate on the counter-flow principle involving sharply increasing wear according as heat recovery rates are higher. Hence it is necessary, with planetary coolers, to find an optimum compromise between the various cost factors: quality of the construction materials, design features, and performance (in terms of lifting and scattering action) of the internal fittings have to be weighed against one another.

The action of some commonly used internal fittings is represented by the curves in Fig. 46. Very poor scattering performance is shown in Fig. 47 (where the scoops are emptied too soon) and Fig. 48 (where they are emptied too late because their discharge openings are not wide enough). On the other hand, good scattering of the clinker in the interior of the cooling tube throughout each revolution of the kiln is shown in Fig. 49. Here the flights or scoops are shaped to a slight twist and have reinforced wear-resistant edges, besides having a number of stiffening diaphragms as a safeguard against buckling.

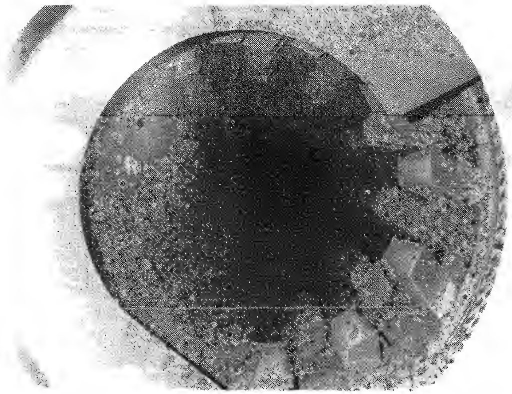


Fig. 48: Lifter flights enclose the clinker too closely and are emptied too late

Fig. 49: The flights are of slightly warped or twisted shape and have strengthened edges to protect them from wear. Besides, there are a number of intermediate stiffening diaphragms to prevent buckling

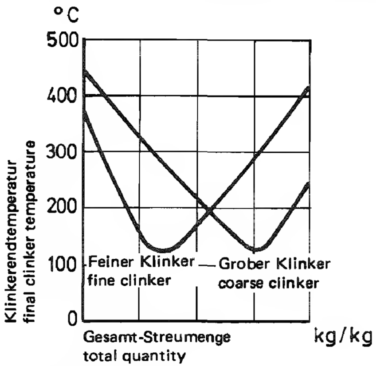


Fig. 50: Clinker exit temperatures

It is to be noted, however, that it is not necessarily always advantageous to scatter the greatest possible amount of clinker. In the case of fine-grained clinker too much scattering action may even be a disadvantage because of the dust cycle it generates. The relationship between clinker exit temperature for various amounts of scatter is shown schematically for coarse and for fine clinker in Fig. 50. It appears that, for fine clinker, the final temperature of the clinker on exit from the cooler becomes higher with increasing proportion of scatter, the reason being that the resulting dust cycle produces the following adverse effects:

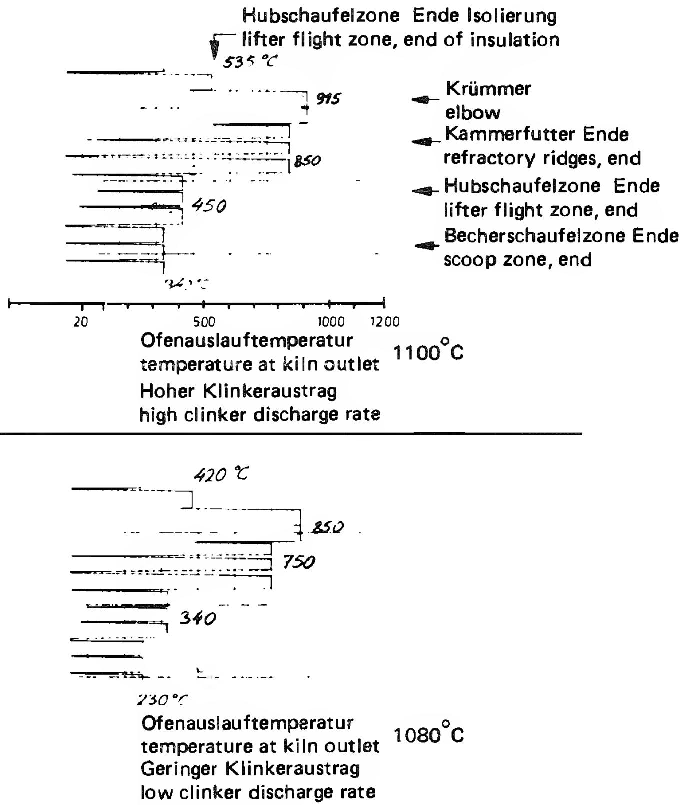


Fig. 51: Cooling curve of clinker

the specific loading of the cooling tubes increases;
clinker which has already cooled is carried back into hot parts of the cooler or indeed into the kiln itself, so that the recuperation efficiency decreases.

As the granulometric composition of the clinker is not always known in advance, optimization of the internal fittings may be possible only when the plant is actually in service. For this purpose it is advantageous to measure the temperature in the interior of the cooling tubes by means of thermocouples. Typical temperature distributions measured in this way, for different operating conditions of the kiln, are represented in Fig. 51.
The grades of material most commonly employed for the internal fittings of the cooling tubes are indicated in Table 5 on page 347. As in the rotary cooler, a cast steel alloy with 30% chromium (material No. 4777) has been found satisfactory. A typical subdivision of the cooling tubes into various zones relating to the types of lining and internal fittings is illustrated in Fig. 52a – g.

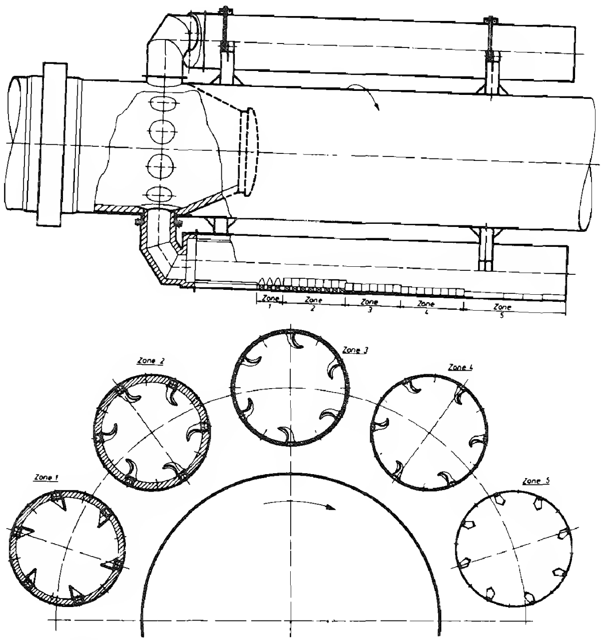


Fig. 52a: Subdivision of the cooling tubes of a planetary cooler (from Herchenbach, 1978)

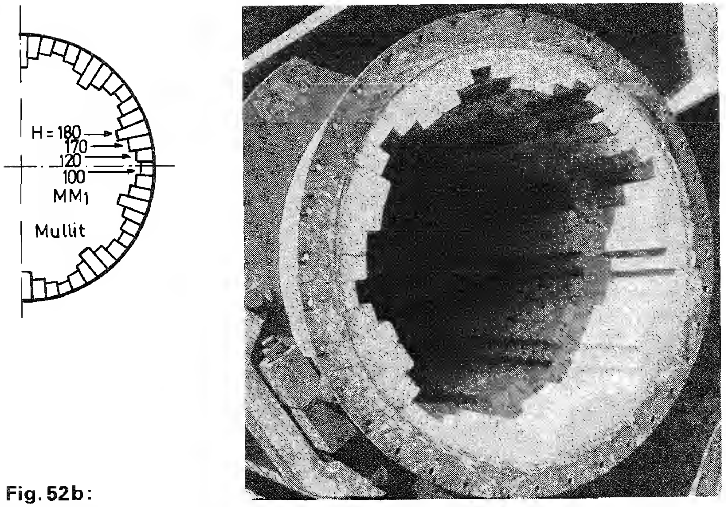


Fig. 52b:

Zone 1: Ridged brickwork (e.g., 6 – 10 ridges 150 – 200 mm high, each formed by three high-alumina bricks of wear-resistant and spalling-resistant grade; intermediate bricks are 100 mm high and of standard hard fireclay grade)

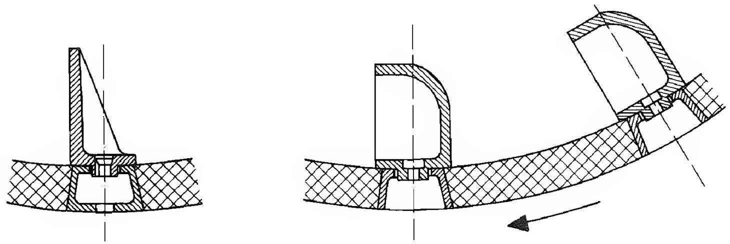


Fig. 52c:

Zone 2: Refractory lining of hard fireclay wedge bricks, 100 mm high, with cast steel breaker teeth set in the brickwork. The object of these teeth is to break up any large lumps of clinker which may enter the cooler when fragments of coating are detached and thus to protect the lifter flights

Fig. 52d:

Zone 3: Refractory lining as in zone 2, but with embedded cast steel lifter flights

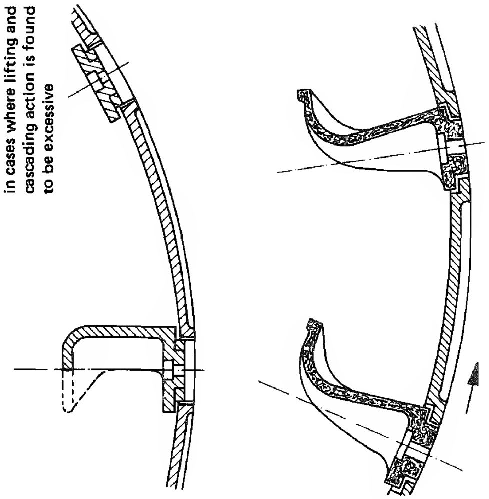


Fig. 52e:
Zone 4: Cast steel wearing plates with heat insulation and cast steel lifter flights

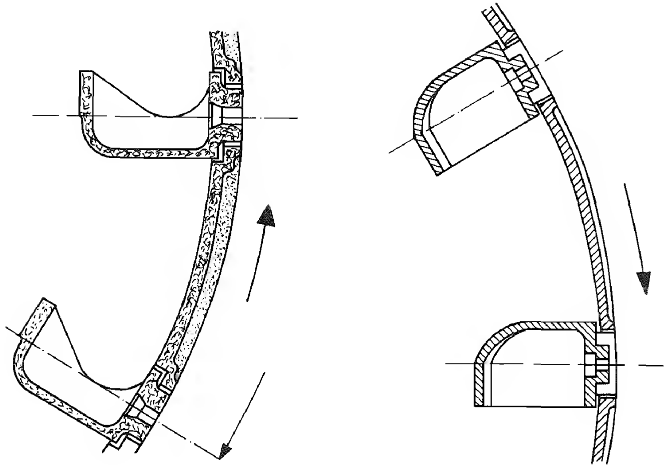


Fig. 52f
Zone 5: Cast steel lifter flights with wearing plates in various forms of construction

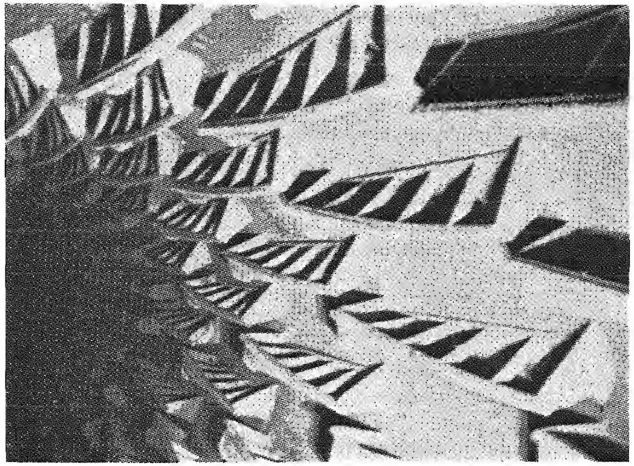


Fig. 52g
Zone 6: Steel scattering flights (from Kadel, 1975)

An example of the optimization of these fittings in a tube of a planetary cooler for a throughput of 3000 t/day is given in Fig. 53.

Heat balance of the planetary cooler

The heat balance for the planetary cooler envisaged in Fig. 53 is set forth in Table 6.

In this example the clinker entry temperature is 1120° C. Although radiation and convection losses are kept low by the thermal insulation provided by the refractory lining and the insulated liner plates, the clinker was cooled to a final exit temperature of about 170° C, without having recourse to water injection. With water injection at a rate of about 3% of the clinker throughput the final temperature was lowered another 40°, which was not attended by any ascertainable increase in heat consumption.

In general, it has been found in practice that planetary coolers installed on heat-economizing kilns should always be backed up by additional cooling facilities in order with certainty to maintain final temperatures below 150° C under continuous operating conditions.

Cooling by the admission of water into the cooling tubes has proved satisfactory for the purposes and presents no problems.

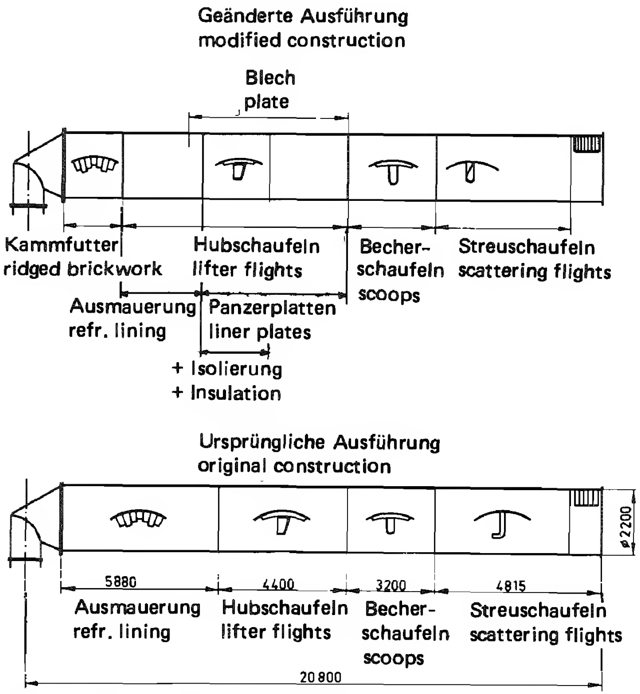


Fig. 53: Optimization of cooling tubes

There are two methods:

(1) Water is sprayed into the outlet ends of the rotating tubes from nozzles mounted on the discharge end housing. A simple control system ensures that they inject water at the correct intervals as the tubes pass the nozzles. See Fig. 54 a.

(2) Water discharged from a circumferential duct, rotating with the cooler, flows by gravity into the tubes. The water is scooped from an open tank, in which the water level can be controlled, and enters the cooling tubes through inlet funnels (Fig. 54 b)

Table 6

kiln type:	preheater kiln
kiln capacity: t/24 h	3000
specific heat consumption	kcal/kg 740
Cooling tube dimensions	
L × D m	2.2 × 19.8
number of tubes:	10
ratio L.D	9
cross-sectional area per tube. m^2	3.81
volume per tube: m^3	75.5
total m^3	755
specific volume rating per day/m^3	3.97
total cooler area: m^2	1365
specific cooler area m^2/per day	0.45
Heat supplied	
clinker + air kcal/kg	264
Heat losses	
clinker kcal/kg	30
radiation + convection kcal/kg	52
secondary air kcal/kg	182
0.9 Nm^3/kg clinker	
634° C	
Thermal efficiency	
cooler %	69
cooler + kiln %	75
power consumption kWh/t	1.5
energy efficiency	
cooler %	66
cooling as a whole %	73

As an alternative to the introduction of water into the cooling tubes, external spraying with water is sometimes applied as auxiliary cooling. This system has various disadvantages, however, besides being more expensive. More particularly, intermittent operation of the spraying system – confining it only to periods when unfavourable operating conditions arise (e. g., excessive clinker discharge from the kiln) – is not possible because the attendant stress variations in the shell plate of the cooling tubes would be harmful to the tubes. For the same reason the quantity of water sprayed onto the tubes should be sufficiently large to ensure that they remain wet throughout a complete revolution of the kiln.

In the example in Fig. 55 the cooling tubes are externally sprayed with water over a length of 7 m, involving heat removal at a rate of 50–80 kcal/kg of clinker. The water is circulated to the spray nozzles at a rate of 100 m^3/hour, the hourly loss

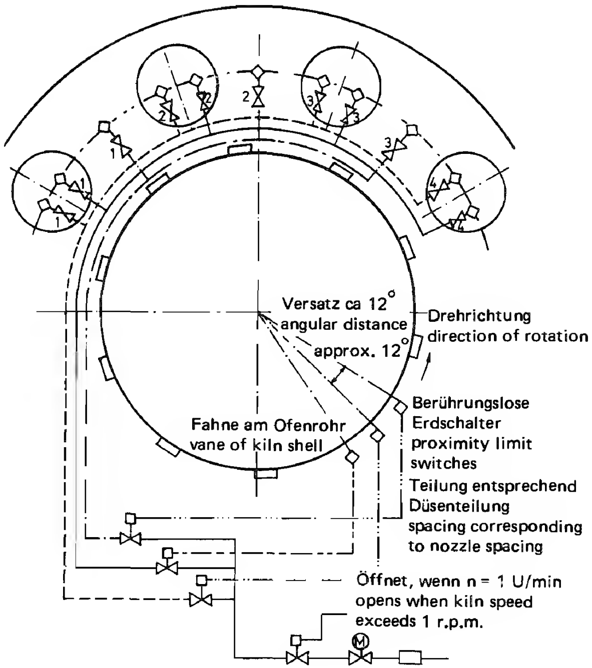


Fig. 54a:

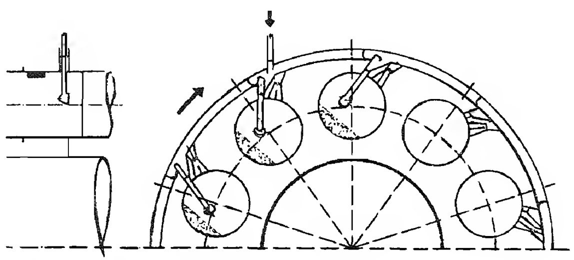


Fig. 54b: Water cooling in planetary cooler (from Duda, 1978)

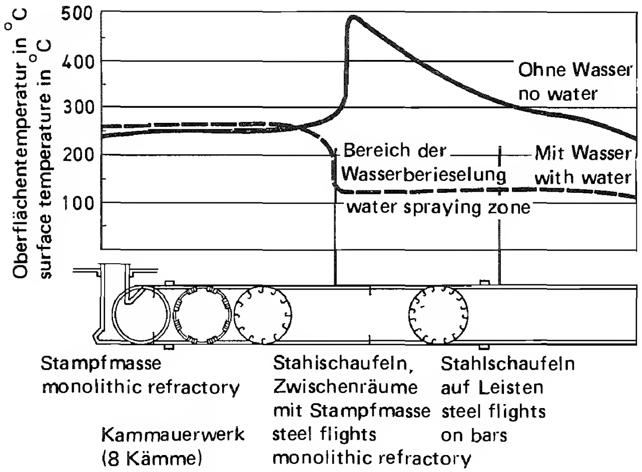


Fig. 55: External waterspraying (from Münk, 1975)

by evaporation being 12–15 m³. Although with this method the clinker exit temperatures can be kept at about 120° C, it is evident, on comparing this with the example for which the heat balance is given in Table 6, that this water cooling is achieved at the expense of the recuperation efficiency and cannot be an economical method. In the present example this method was nevertheless adopted for reasons of environmental protection against noise nuisance. For the sake of noise control, part of the planetary cooler – the zone comprising the lifting scoops – had to be provided with a sound-attenuating enclosure. The problem of getting rid of the heat trapped in this acoustic hood was solved by external water spraying. See Fig.56.

3.3.3 Rotary coolers

The rotary cooler is the oldest type of clinker cooler built to operate in conjunction with rotary kilns. With the introduction of the modern heat-economizing kilns, however, it has largely fallen into disuse. In new plants its use is confined to a few special cases.

The cooler consists of a drum or tube inclined at an angle of 4–7 degrees, supported at two points along its length and rotated by means of a girth gear and pinion drive which can be controlled, independently of the kiln, to give speeds in the range of 0.3 to 3 r.p.m.

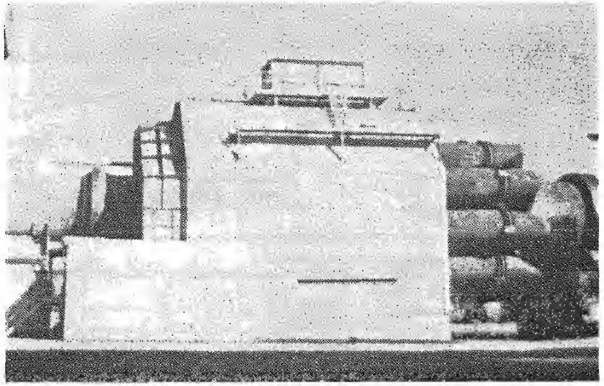


Fig. 56: Movable acoustic hood to suppress noise emission (from Munk, 1975)

The hot clinker falls directly from the kiln into the cooler, in which its movement is achieved by the slope and rotation of the tube and with the aid of internal fittings. As a result of the negative pressure existing in the kiln, cold air is drawn in from the open (outlet) end of the rotary cooler. This air flows through the cooler and cools the clinker To achieve optimum heat exchange, the air flow velocity must not be too low. On the other hand, it must not be too high either, otherwise it will tend to obstruct the movement of fine clinker particles down the cooler and thus cause congestion.

The following design dimensions have generally been found satisfactory·

throughput: 2.2 – 3.0 t of clinker per day and per m^3 of internal volume of cooler; length/diameter ratio: 10:1 to 15:1.

For the design of the drive, tyres (riding rings), rollers, rotating seals and wall thicknesses the principles and criteria are basically the same as those for the rotary kiln itself. In view of the risk associated with overheating of the shell plate, the latter should be at least of a boiler plate grade.

About 70% of the length of the rotary cooler is lined with refractory material, generally in the form of fired brick, graded from alumina (sometimes high-alumina) brick in the hot zone to semi-acid fireclay brick at the cooler end of the refractory lining.

Embedded in the refractory brickwork are lifters made of heat-resisting and wear-resisting cast steel. Purely chrome-alloy steel grades with about 30% chromium content have been found suitable for the purpose, the more so as they are relatively inexpensive. Ceramic internal fittings are not suitable for rotary coolers. Nor have special lifter bricks proved satisfactory, as their heads spall or wear down too rapidly. Scoops (or flights) made of wear-resisting steel and designed to scatter

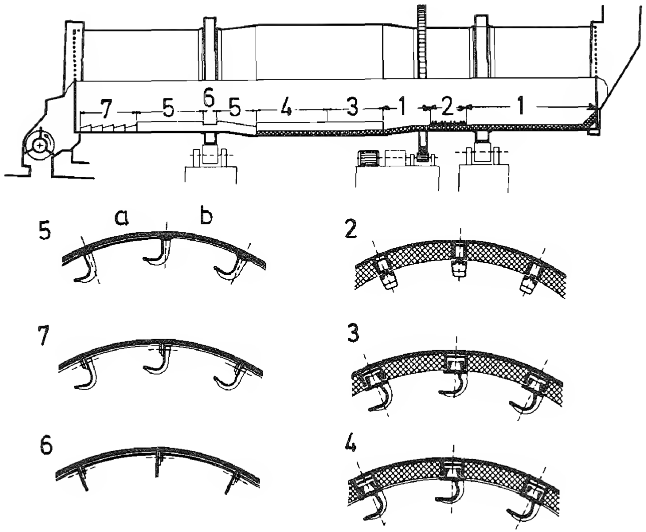


Fig. 57:
Rotary cooler – internal fittings (according to Herchenbach, 1978)

the clinker are installed in the after-cooling zone. Some typical fittings installed in rotary coolers are shown in Fig. 57. The following are to be distinguished.

refractory-lined zone without ridges (1); refractory-lined zone with cast steel breaker teeth (2); refractory-lined zone with lifters (3); refractory-lined zone with lifters and additional wearing plates (4); lifters with wearing plates (liners) (5); tyre zone with wearing plates and lifter bars (6); scattering flights with wearing plates (7).

Power consumption of rotary coolers is low For such a cooler designed in accordance with the criteria described here, the following formula is valid:

$N_W = f_K \cdot L \cdot D^2 \cdot n$

where.

N_W = power at the motor shaft (kW)
L = length of the cooler (m)
D = diameter of the cooler (m)
n = speed of the cooler (r.p.m.)
f_K = factor ranging from 0.09 to 0.13 (0.09 for coarse clinker, 0.12 for fine clinker and large number of scoops)

Because of the high starting torque and the need for reserve drive power capacity to cope with overload conditions, the installed motor power rating should be at least 30% higher than the figure calculated with this formula. It should further be taken into account that with a rotary cooler, as with a planetary cooler, the power consumption of the exit gas fan will be higher than for a kiln with a grate cooler.
Since the cooling air rate in the rotary cooler is predetermined by the amount of secondary air that the kiln can usefully accept, the cooling effected in such a cooler installed behind a heat-economizing kiln is not sufficient to cool the clinker to an acceptably low exit temperature. Results can be improved in this respect by spraying water on the outside of the cooler in the after-cooling zone. A simpler method, however, is to spray water into this zone of the cooler. So long as the amount of water thus injected does not exceed about 30–50 g/kg of clinker, it does not have any discernible adverse effect in terms of process engineering performance. Problems in the operation of the rotary cooler may be caused more particularly by dust cycles due to high air velocities and a high content of fine-grained clinker. In such cases it may be necessary to increase the diameter of the tube in the critical zone where the lifters are installed.
Another part where problems are liable to arise is the inlet chute on which accretions ("snowmen") are formed, especially in large installations with a considerable height of fall of the hot clinker. Remedial measures consist in water-cooling the chute or equipping it with an automatic dislodging device.

3.3.4 Shaft coolers

In the present state of the art the shaft cooler remains suitable only for very small units and for plants with exceptionally favourable raw material conditions which guarantee uniform particle size distribution of the clinker with only small proportions of coarse and fine particles and with a constant rate of clinker discharge.
The shaft cooler is purely a counter-current cooler. The clinker falls into a vertical cylindrical shaft and makes its way downwards to the outlet from where it is extracted through a grate comprising a number of breaker rolls. Its movement through the shaft is similar to that of the material in a shaft kiln. See Fig. 58.
In the Walther-Beratherm shaft cooler the upper part of the shaft is of reduced diameter in order to increase the cooling air flow velocity in this part and thus produce a fluidized bed effect with the object of distributing the incoming clinker (discharged from the kiln) over the whole shaft cross-section and improving the heat transfer.
Air consumption for cooling is about 1.05–1.1 Nm^3/kg of clinker. About 35% of this air is introduced under the grate, 45% into the middle part of the shaft, and 20% into the narrower upper part. Air distribution over the shaft cross-section is achieved through specially designed nozzles and air tubes extending into the clinker mass itself.
Thermal efficiencies from 75 to 80% and upwards can be attained. The advantage of the high rate of heat recovery is, however, partly offset by additional exit gas heat losses due to the greater amount of secondary air and the high specific power

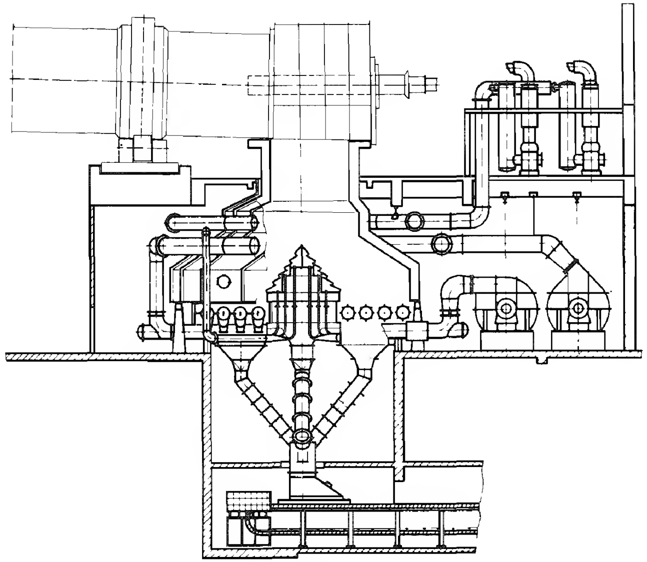


Fig. 58: Shaft cooler (from Herchenbach, 1978)

consumption, amounting to 8–10 kWh/t of clinker and necessitated by the very high static pressures of over 110 mbar for which the cooling air fans have to be designed. The clinker exit temperature ranges from 250° C to over 350° C, so that after-cooling of the clinker or suitable cooling arrangements in conjunction with clinker grinding are essential.

3.3.5 Gravity coolers

The gravity cooler, or "g" cooler, can serve only as an after-cooler for dealing with clinker which has already been cooled to about 500° C and in which the coarser lumps have been crushed to a size that the cooler can accept.
The clinker is distributed by a drag-chain and descends through the cooler by the action of gravity alone. On its way down it does not come into direct contact with the cooling air, but slides slowly in a densely packed mass past the cooling tubes, of flattened lenticular cross-section, through which the air flows. The air is blown into the cooler from below and makes its way upwards through successive banks of tubes, leaving the cooler at the top (Fig. 59).
The final (exit) temperatures attained by the clinker are between 50° and 100° C. Air consumption is in the range of 1.2–1.8 Nm^3/kg of clinker. As the pressure drop

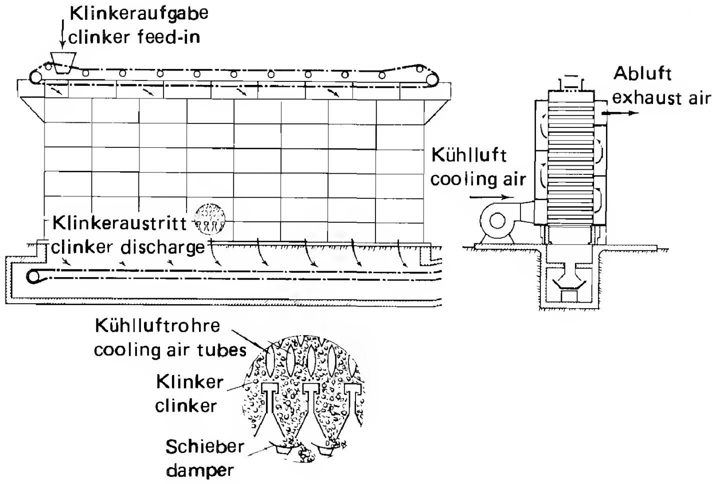


Fig. 59: "g" cooler (from Steinbiß, 1972[1])

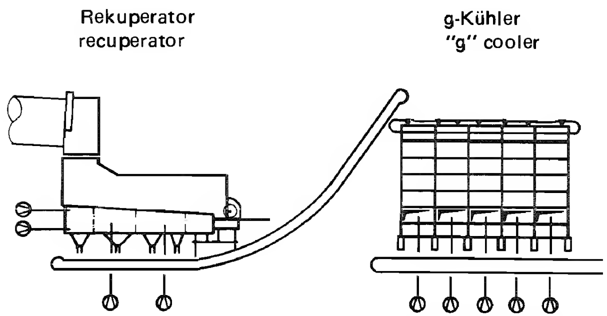


Fig. 60: Combination of recuperator with "g" cooler (from Hellberg, 1977)

is only about 80–120 mm w.g., the specific power consumption is fairly low: in the region of 1 kWh/t of clinker.
To achieve adequate heat transfer, the clinker has to move through the cooler at a low speed (1–2 m/minute), so that its retention time in the cooler is relatively long (2–3 hours) and the rate of wear is very low. Depending on the cooling range required, design is based on a throughput of 9–11 t of clinker per day and per m^3 of volume of the cooler.
For new plants in which there is no scope for utilization of the exhaust air from the cooler, the gravity cooler is installed directly behind a reciprocating grate cooler which functions purely as a recuperator (Fig. 60). In that case the air supplied to this grate cooler must be accurately equal to the required secondary air, and under such conditions the exit temperature of the clinker discharged from the grate cooler will on average be about 750 K. The clinker breaker will have to be equipped with a suitable cooling system, as already described with reference to Fig. 26.
Apart from operating behind a reciprocating grate cooler functioning purely as a recuperator, a gravity cooler can sometimes advantageously be installed behind a reciprocating grate cooler with exhaust air utilization, more particularly in a case where the kiln output has been subsequently increased and the existing grate cooler is no longer able to achieve the required final clinker temperature. The example illustrated in Fig. 61 relates to such a case where the capacity of the kiln was increased from 1650 t/day to 2000 t/day. The design and operating data for the coolers are schematically represented in Fig. 62.

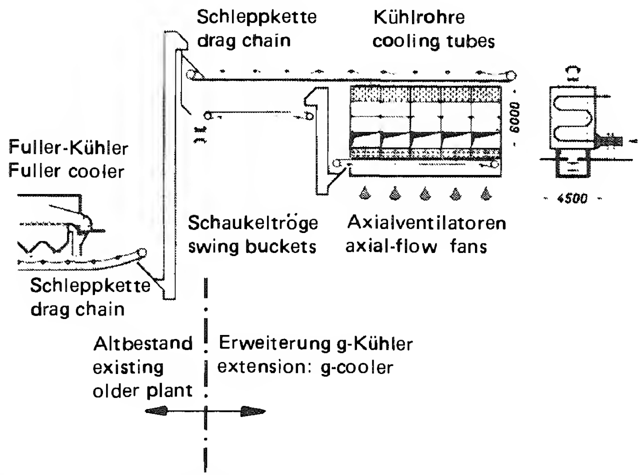
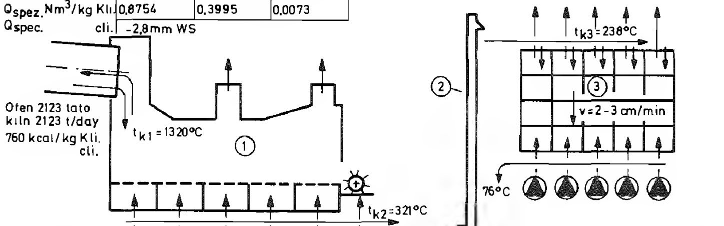


Fig. 61: "g" cooler (from Kwech, 1974)

	Sekundärl. secondary air	Mittenluft centre exit a	Abluft exhaust air
	t_S = 860 °C	t_M = 380 °C	t_A = 320 °C
Q Nm³/h	77,423	35,333	680
$Q_{spez.}$ Nm³/kg Kli. / $Q_{spec.}$ cli.	0,8754	0,3995	0,0073
	-2,8 mm WS		

Austritt exit	t °C	44	49	59	68	88



Kammer compartment	1	2	3	4	5	Falschluft inleaked air	Summe total
t_L °C	+ 35	+ 15	+ 15			+ 15	
mm WS	430	360	261	1			
Q Nm³/h	32.800	31.400	46 900			2.317	113.417
$Q_{spez.}$ Nm³/kg Kli. / Q_{spec} cli.	0,37	0,353	0,533			0,0262	1,2822

Eintritt inlet						Summe total
t °C	+ 4	+ 4	+ 4	+ 4	+ 4	
Q_{gk}	39800	36400	28900	26100	18100	149300
$Q_{gk\,sp}$	0,453	0,41	0,327	0,295	0,205	1,69
kW	17	17	16	16	15	81

Fig. 62: Operating data for the combination considered in the design example

1 Fuller cooler: η thermal = 68.2%
2 clinker handling: η heat loss = 29.5%
3 gravity cooler: η heat dissip. = 70.7%

Power consumption:		
	gravity cooler fans	81.0
	feed drag chain	13.0
	extractor belt	4.5
	rocker troughs	2.2
	total	100.7 kW

Specific power consumption 1.14 kWh/t of clinker

3.4 Operation, monitoring, measurement and control of coolers

3.4.1 General considerations

The cooler is an integral part of the clinker burning process. Kiln and cooler interact and have to be adjusted to each other in their manner of operation. For optimum performance of the process as a whole the cooler should aim at attaining.

- consistently high secondary air temperature;
- low clinker exit temperature.

The kiln should be so operated as to attain:

- uniform clinker discharge;
- uniform clinker particle size distribution;

- constant temperature of the burning zone;
- unvarying combustion air conditions.

Irrespective of the type of cooler, the following are important measured variables and controlled variables for monitoring the operation of the cooler:

- the temperature of the clinker at the kiln outlet;
- the secondary air temperature;
- the final temperature of clinker on leaving the cooler.

For the particular types of cooler there are moreover other important parameters. The number of variables to be measured and controlled is greatest in the case of grate coolers, for which, besides those listed above, the following have to be measured:

- the cooling air rate supplied by the fans;
- the pressures in the undergrate compartments;
- the exhaust air rate and temperature;
- the kiln hood pressure.

Another very useful aid is television monitoring of the bed of clinker on the grate, including the inlet chute.
Proper monitoring of the mechanical functioning of the cooler requires:

- temperature measurement at the surface and within the material of components especially at risk;
- speed and function monitoring of moving parts;
- current and power consumption measurement;
- interlocking of material handling sequences.

3.4.2 Grate coolers

A grate cooler comprises a large number of individual drives, requiring considerable measuring and control equipment to ensure reliable functioning under optimum conditions.
The measured and controlled variables involved will be explained with reference to examples of combination coolers with and without duotherm air operation. See Fig. 63.
In the main, the variables which are measured and recorded are temperatures, pressures, flow rates and rotational speeds, comprising more particularly:

secondary air temperature
exhaust air temperature
clinker exit temperature
temperature of grate plates
pressure in kiln hood
undergrate pressures
reciprocating grate movement
cooling air rates.

Except for the measurement of the secondary air temperature, the above-mentioned quantities can all be measured with standard detecting elements.

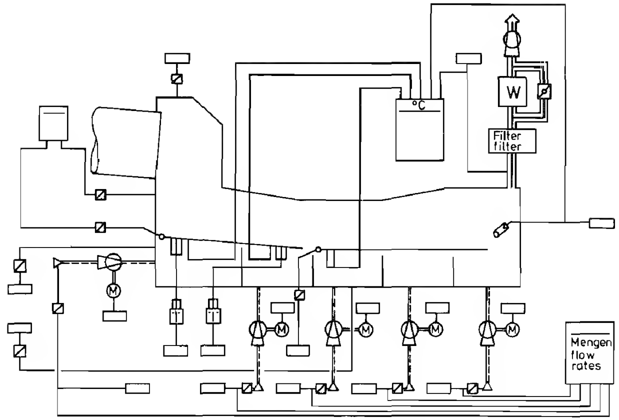


Fig. 63: **Diagram of measuring instrumentation**

The main reason why the secondary air temperature is difficult to measure is that the temperature field in the feed shaft of the cooler is mostly inhomogeneous. Thus, it is by no means unusual to obtain simultaneous measured values ranging from 300° to 1000° C at different measuring points and with different methods. Ordinary commercial thermocouples inserted laterally or from above into the rising secondary air are indeed exposed to the temperature of this air stream, but also receive much radiation from the incandescent clinker and thus give readings which are generally much too high. To overcome this problem, suction thermocouples have been devised which are provided with a concentric outer tube around the temperature sensor. The secondary air is continuously sucked into the tube by means of a jet pump worked with compressed air Even so, a suction thermocouple can sample only a limited portion of the air flow and moreover requires much maintenance and attention. Since it is not possible to perform accurate measurements on the secondary air, even with elaborate instrumentation, many plant operators content themselves with determining only a relative value and trends. A simple method requiring little maintenance of equipment consists, for example, in using a radiation pyrometer which is mounted on one side of the feed shaft and is aimed at the opposite wall. Control itself is based on substitute variables, usually the pressure in the first cooling air compartment.

Grate cooler controls

Fig. 64 schematically shows the commonly employed control circuits for a combination cooler with duotherm air. The purpose of the control system is to

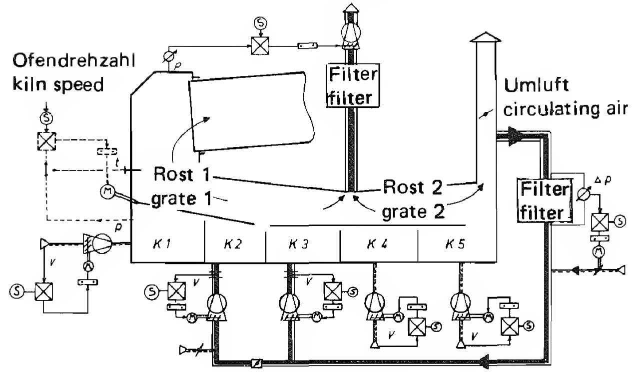


Fig. 64: **Control loops for the Fuller cooler**

enable the cooler to achieve its optimum recuperation efficiency by as nearly automatic adjustment as possible. To achieve this, the kiln must at all times receive sufficient secondary air with the highest attainable constant temperature. In addition, the exhaust air extracted intermediately along the cooler and intended for utilization of its heat content should have a high temperature, while the exit temperature of the clinker on discharge from the cooler should be low.
These requirements imply that the control system should achieve optimum air distribution and favourable clinker bed depth.

Cooling air rates

As the cooling air is supplied by a number of fans, the respective proportions supplied by each of them should remain constant in order to maintain the desired air distribution. The air flow rates must be maintained irrespective of the varying flow resistance through the bed of material on the grate. In the case of the five-compartment cooler envisaged in the example there are five individual control circuits. Flow rates are measured either at inlet nozzles or, for the warm air fans, by means of venturi-type constrictions in the air duct. In the event of deviations from a preset value, the inlet control vanes on the fan concerned are adjusted to compensate for them.
The warm air fans are protected from excessively high temperatures by control of a damper admitting cold air into the system. Similar control arrangements for the introduction of external air and for water spraying are provided for protecting the exhaust air dust collecting equipment under critical temperature conditions.

However, in the present example such sensitive equipment is not necessary, adequate dedusting being achieved in large cyclone collectors which serve only to protect the fans from excessive wear and which are not affected by adverse temperatures.

Kiln hood pressure

The rate of flow of the secondary air from the cooler to the kiln is indirectly stabilized with the aid of the pressure in the firing hood of the kiln.
Because of temperature and flow conditions in the hood, the pressure not only varies from one measuring point to another, but also pulsates very considerably. The usual method of coping with this consists in averaging the pressure at the side and at the roof by means of a ring duct, which moreover damps the fluctuations. In large plants the difference in kiln hood pressure as measured at the side and at the roof may amount to several mm w.g. This being so, even with a properly functioning kiln hood pressure control system and a set point of ± 0 mm w.g. in the upper part of the hood, a certain amount of dust-bearing hot air is bound to escape, while external ("false") air will infiltrate into the lower part if the kiln hood seal is not completely effective.
The hood pressure control system functions as follows: When the pressure in the hood rises, which may occur for example as a result of a decrease in the exit gas flow or an increase in the cooling air flow, the controller increases the volume of air delivered by the exhaust air fan. It does this by adjustment of a damper or an inlet vane control unit or by varying the fan drive motor speed. Conversely, when the pressure in the hood goes down, the air delivery rate of this fan is reduced by the control system. Thus, with the aid of the hood pressure controller, the exhaust air fan performs the function of a pressure relief valve.

Secondary air temperature

The pressure in the first cooling air compartment is used as a substitute variable for the secondary air temperature which, as already explained, cannot be reliably measured. This pressure is kept constant by means of a controller which regulates the movement of grate 1.
Another control circuit maintains a constant speed ratio of grates 2 and 1. For this purpose either the speeds of these two grates are controlled direct or control is based on the pressure in the first compartment under grate 2.
As the control of the secondary air temperature via the undergrate pressure is not entirely straightforward, a number of interrelationships have to be taken into consideration. The negative pressure is affected chiefly by the following factors:

(1) depth of the clinker bed;
(2) granulometric characteristics of the clinker;
(3) distribution of the clinker on the cooling grate;
(4) temperature of the clinker and cooling air;
(5) cooling air supply rate.

To start with, it must be presupposed that the cooling air flow rate control system, described earlier on, is functioning properly. If the set point (desired value) for the cooling air rate is changed, the set point for the pressure in compartment 1 must also be altered. If a process control computer is used, these adjustments will be made automatically. Besides, with a computer, it has been found advantageous to incorporate a disturbance compensation system based on control of the rate of clinker discharge from the kiln. With this arrangement the quantity of clinker fed to the cooler per unit time is kept reasonably constant by controlled adjustment of the kiln speed. This adjustment has to be performed very sensitively, however. The desired value of the kiln speed is so calculated with reference to the pressure in compartment 1, the cooling air supply rate to compartment 1 and the reciprocating frequency of grate 1 that it varies on average by only a small amount on either side of a certain desired kiln speed depending on the overall clinker output to be attained. In this way, despite short-term control of the discharge rate, effective long-term control of the kiln loading factor and clinker production is obtained.
Furthermore, by taking account of the various influencing parameters with the aid of the computer, it can be decided whether the fluctuations in clinker discharge from the kiln are to be rated as normal, so that the system can operate with the normal setting of the controller, or whether the fluctuations are abnormally large, e.g., due to surges of dislodged coating, and require extremum control for their correction. From these comments it will be evident that conventional pressure control is problematical and that the control desk operator should always keep a watchful eye on the clinker cooler and be ready to intervene even if the cooler is being run under automatic control. Such intervention may also be necessary from time to time for the sake of optimization, as the desired value of the pressure will have to be re-set in response to possible changes in the granulometric characteristics and in the distribution of the clinker (e.g., due to ring or coating formation at the kiln outlet). These adjustments may be applied with the aid of a computer which automatically integrates the heat losses, establishes heat balances for the cooler and determines the recuperation efficiency. Further important decision criteria are the grate plate temperatures and the television image of the clinker bed and clinker discharge from the kiln. The size of compartment 1 is also important in connection with control and should preferably not comprise more than five plate rows, so as to achieve a rapid response to changes in the clinker bed.

3.4.3 Rotary and planetary coolers

With rotary and planetary coolers the secondary air temperature can hardly be influenced by the mode of operation of the cooler. Elaborate control arrangements are therefore unnecessary, and the principal object of the measurements is to give prompt warning of critical operating conditions in order to prevent mechanical damage.
The surface temperature of the cooling tubes is therefore the most important measured variable, which is determined with a radiation pyrometer or by infrared television connected to a mini-computer, so that, besides a grey scale thermal diagram, the temperatures are made directly "visible" and abnormally high temperatures are promptly detected (Figs. 65a and 66).

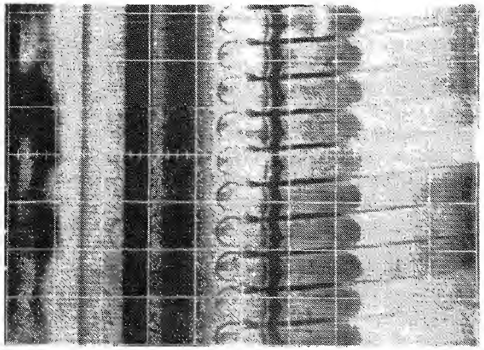


Fig. 65a: Grey scale thermal diagram for planetary cooler

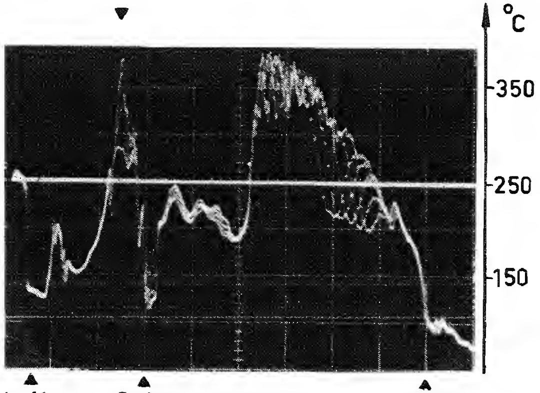


Fig. 65b: Shell temperature of a cooling tube, infrared measurement

In the event of critically high shell temperatures of a temporary character, large cooling air fans – permanently installed under the cooler in many installations – can automatically be switched on when needed. On the other hand, any water spraying applied to the outside of the cooling tubes must operate continuously. Intermittent operation of the sprays, or even too great a reduction of the water flow rate (so that the tube surfaces are not kept permanently wet), is liable to cause damage to the shell plate in consequence of thermal stresses. Another critical point in planetary cooler systems is the kiln shell outlet section at the clinker discharge ports. The stress conditions are particularly severe in this region, where cracking of the shell and serious damage to the kiln can very quickly occur in the event of excessive temperature. To cope with abnormal stress conditions it is essential, besides providing a maximum-value (400° C) monitoring and warning system, also to monitor the temperature difference between adjacent discharge ports. This difference between any two ports should not exceed 70° C. Since the critical temperature range is difficult to ascertain from a lateral measuring position, it is advisable to install a separate radiation pyrometer which is aimed obliquely from below at the planetary cooling tubes and the outlet section of the kiln shell.

The clinker exit temperature, i. e., the temperature at which it is discharged from the planetary or rotary cooler, may sometimes become too high, e. g., in the event of abnormally high clinker output from the kiln due to dislodgment of coating. For this reason the clinker temperature is usually measured directly after the cooler, by means of a radiation pyrometer, and the rate of cooling water supply to the cooler is controlled accordingly. Measurement of the clinker temperature with a radiation pyrometer, however, has the disadvantage that only the surface temperature is determined and that the temperature readings fluctuate considerably. In order to connect the pyrometer to an automatic control system it is therefore necessary to connect a strongly damping integrator in the output of the measuring device.

In some cases it has been found advantageous to measure the clinker temperature inside the cooling tubes of planetary or rotary coolers. The clinker temperature differentials in the zone equipped with refractory lifting ridges are of especial interest. They provide a good indication of the clinker discharge from the kiln and can be utilized for disturbance compensation in controlling the fuel feed rate.

Transmitting the temperature measurements obtained with thermocouples is somewhat elaborate, with slip-ring pick-up and automatic switch-over from one measuring point to another (Fig. 66) or with telemetric transmission. As a rule, however, the temperature measurements for the cooler can be combined with the kiln shell temperature monitoring at the adjacent kiln tyre, in which case the extra expenditure involved is very little.

The operation of planetary and rotary coolers requires no special attendant personnel for control. The cooling air flow rate, the clinker discharge rate and the secondary air temperature automatically adjust themselves in relation to the clinker output of the kiln, the heat consumption and the temperature of the clinker on entering the cooler. Fluctuations in the discharge rate or temperature of the clinker on leaving the kiln which are caused by, for example, dislodgment of coating cannot be compensated in the cooler

In a kiln with planetary cooler the clinker discharge ports and the distribution of the

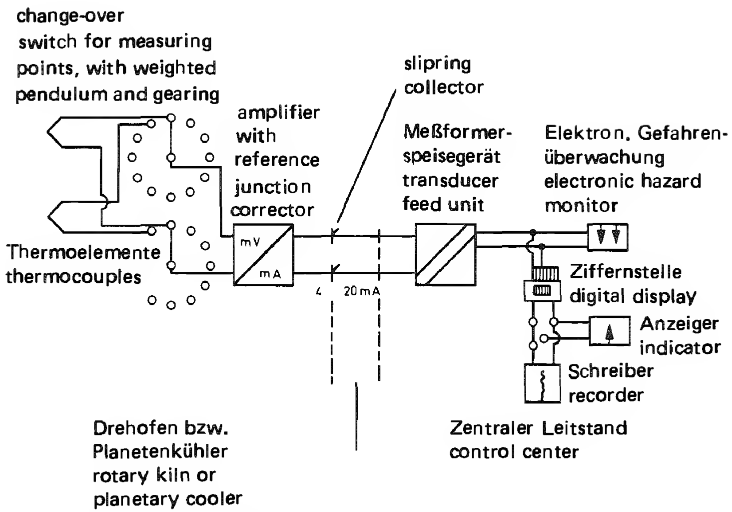


Fig. 66: Diagram of measuring system for clinker temperatures in the planetary cooler

clinker over the individual cooling tubes are critical. If the material getting into the ports is too hot and sticky, they can become choked; also, large pieces of detached coating are liable to become wedged in the ports and cause congestion in the kiln.
Another source of uncertainty in the cooling process arises from differences in the quantities of clinker discharged into the respective cooling tubes, some of which will receive more and others less clinker, depending on coating conditions in the kiln and on the degree of wear at the kiln outlet. As a result, cooling will take place at a slower rate in the over-filled tubes, while the internal fittings in these will be subjected to heavier loads and rougher treatment.
Another drawback of the planetary cooler is that fluctuations in the clinker discharge from the kiln cannot be evened out in the cooler. The consequences are apparent from Fig. 67.
These curves were plotted from a kiln test in which the plant was operated with a constant rate of raw meal feed and constant kiln speed.
The top diagram indicates the variations in clinker output, corresponding to hourly values ranging from 2600 to 3400 t/day, while the short-term fluctuations are even greater. The ten-minute integral varies from 2000 to 4000 t/day. The clinker handling system must reliably cope with these substantial surges in the clinker

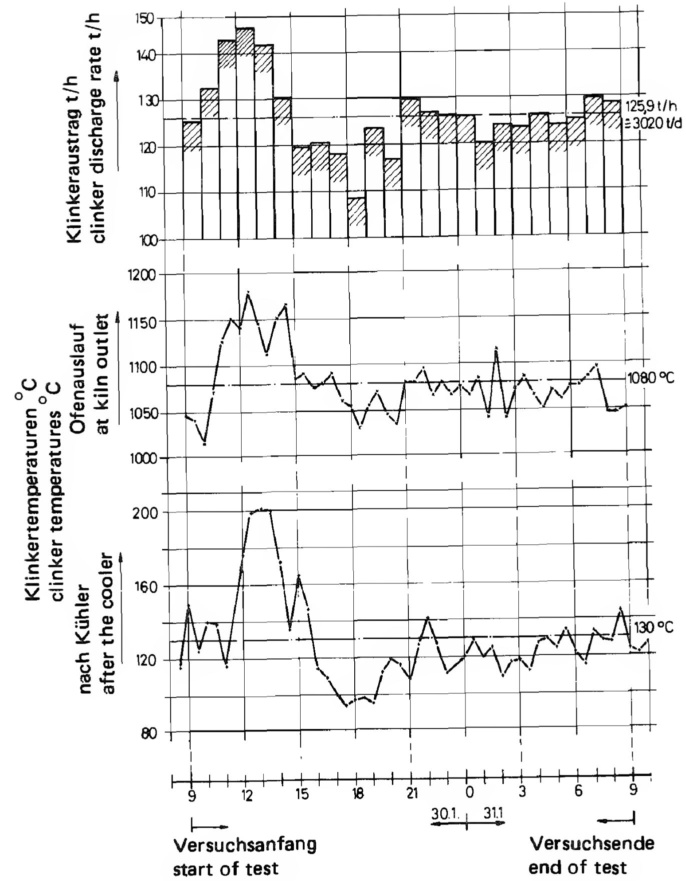


Fig. 67: Clinker discharge rate from kiln and clinker temperatures

discharge rate. The other two curves represent the corresponding clinker temperatures at the kiln outlet and on discharge from the cooler. The interaction that occurs is manifest.
A high clinker temperature at the kiln outlet inevitably results in high temperatures of the clinker discharged from the cooler. In addition, the temperature of the

secondary air rises. Burning conditions in the kiln become "harder". On the other hand, it is troublesome to restore a kiln from an under-burning to a hard-burning operating condition. In such a kiln the secondary air temperature is low and, in addition, a considerable dust cycle is established between the cooler and the pre-cooling zone in the kiln. This dust cycle causes a further lowering of the temperatures.
The dead time and the time constant of the system are large. Stable operation, more particularly in a kiln plant with planetary cooler, therefore requires that disturbances are compensated already before the burning zone.
The residence time in the cooler envisaged in the example is about 30 minutes for clinker of 10 mm average particle size and for a kiln speed of 2.1 r.p.m. An unfavourable phenomenon is that large pieces travel faster and that small particles travel more slowly through the cooler. If the clinker has a high content of fines, with a substantial proportion of particles under 1 mm, objectionable dust cycles are liable to develop.
The sound levels emitted by the lifter zone change quite distinctly with variations in the running of the kiln and often correspond (with some time lag) to the variations in the burning zone temperature. With a high fines content the sound levels, measured at a distance of 1 m from the cooler, may vary by about 20 dB(A). If these variations in sound are utilized as a criterion for kiln control, it should be remembered, however, that a high proportion of fine particles may, under certain raw material conditions, also be formed as a result of "over-burning" the clinker.

3.4.4 Shaft coolers

The measuring and control instrumentation for a shaft cooler is somewhat less elaborate than that for a grate cooler. The control duties to be performed are similar, however. The column of clinker in the cooler must be maintained at constant height irrespective of the rate at which clinker is discharged from the kiln.
At the same time the rate of air supply to the cooler must be adjusted to the clinker discharge rate, while the kiln hood pressure should remain as nearly constant as possible. In addition, the clinker exit temperature is controlled by water spraying.

Control of cooling air or secondary air flow

The cooling air flow rate is measured at inlet nozzles on the fans. If changes in pressure gradient through the clinker bed occur as a result of changes in the granulometric composition of the material, the air delivery rate of the fan will also change, and so will the pressure differential at the nozzle. The setting of the inlet vane control unit of the fan will then be altered until the desired value for the cooling air flow rate is restored. This desired value, however, is not itself a constant quantity, but is subject to feedforward correction by the kiln hood pressure, so that, in the event of a change in the exit gas flow rate, the cooling air flow rate is automatically adjusted and the infiltration of "false" air at the hood is kept to a low value.

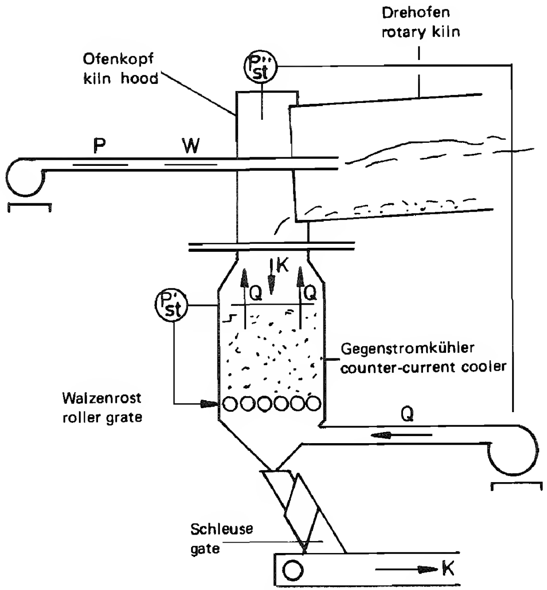


Fig. 68: Shaft cooler with control scheme (from Bade, 1969)

Clinker column control

The excess pressure in the column of clinker over the crushing roller grate is measured with a heavily damped measuring transducer and compared with a reference value. A controller then adjusts the speed of the discharge rollers as a function of the deviation from that value. The problem factor in this control system lies in the granulometric characteristics of the clinker, just as it does in controlling the depth of the clinker bed in the reciprocating grate cooler. Changes in fineness of the clinker particles cause major changes in the pressure distribution and in the heat transfer. As a result of these phenomena, very high clinker exit temperatures may occur; after-cooling facilities for the clinker are therefore essential.

Final temperature of clinker

A water spraying device under the grate serves to lower the final clinker temperature, which is measured by means of thermocouples at the bottom of the shaft. A controller changes the setting of a valve in the return flow pipeline of the return-flow nozzles.

3.4.5 Gravity coolers ("g" coolers)

As the cooler only has to handle clinker of relatively low temperature and is fed with pre-crushed material, its operation presents no problems.
All that is needed is a system for controlling the level of the material in the cooler. This is measured with the aid of gamma radiation and kept constant through a system which controls the functioning of the vibrating trough type equipment for discharging the cooled clinker.
The rate of air delivery by the cooling fan can be adjusted by manual control of the inlet vanes. Such adjustment, however, is necessary only in the event of major changes in the rate of clinker output from the kiln. If the kiln is to be run at reduced output for a long period, one or more compartments of the gravity cooler can be shut off in order to save electric power consumption.
Table 6 gives some performance figures of a gravity cooler installed directly behind a reciprocating grate cooler functioning purely as a recuperator. With this arrangement no exhaust air is diverted from the cooler. Hence the control of the grate cooler must ensure that the rate of cooling air supply to this cooler is exactly equal to the secondary air demand of the kiln. For this reason the kiln hood pressure is kept constant by varying the set point for the air delivery rate of the last cooling air fan of the grate cooler.

Table 6

measuring point	measured values for rated output 2000 t/day, in °C		
	normal	maximum	minimum
clinker exit from kiln	1350	1400	1325
clinker exit from recuperator	520	750	200
clinker entry into "g" cooler	510	730	193
clinker exit from "g" cooler	113	130	50

3.5 Dust collection arrangements for clinker coolers

3.5.1 General considerations

Separate dust collection systems are needed only for grate coolers in so far as there is a surplus of exhaust air and no after-cooling in a gravity cooler is employed. The exhaust air design conditions for dust collectors intended for operation with

clinker coolers of heat-economizing kiln plants are generally within the following ranges:

specific exhaust air rate: 0.7 – 1.8 Nm^3/kg of clinker;
exhaust air temperature: 200° – 400° C;
dust content in exhaust air: 0.7 – 15 g/Nm^3;
proportion of dust particles <10 microns: 0 – 20%.

The dust content of the exhaust air and the fineness of the dust depend greatly on the granulometric composition and the degree of burning of the clinker. The dust content is likely to be near the lower end of the range in the exhaust air from clinker coolers installed behind Lepol kilns, whereas it is likely to be near the higher end in the exhaust from those installed behind kilns with preheater equipment.
The following types of equipment are used for dust collection from clinker cooler exhaust air:

- centrifugal dust collectors;
- granular bed filters;
- fabric filters (preceded by air-to-air coolers);
- electrostatic precipitators.

In comparing the various types of dust collecting equipment with a view to making a choice it is necessary to consider not only the collection efficiency they attain, but also the capital cost and operating expenses.
Capital cost comprises, in addition to the cost of the actual dust collector, the following items:

- the air cooler (e.g., of the air-to-air type) or water spraying system;
- the fan with motor;
- the control equipment;
- the high-tension equipment;
- the dust discharge system with motors;
- the filter cloths or the granular bed packing;
- the electrical installation and erection of the component units.

Operating expenses comprise:

- the electric power consumption of the filter (pressure drop),
- the high-tension equipment,
- the motors for dust handling;
- the maintenance and repair costs;
- the cost of spare parts.

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4 Firing technology

By E. Steinbiss

4.1 Fuels

The fuels commonly used for the burning of cement clinker are listed in Table 1. The values given in this table are averages.

4.1.1 Coal

As a rule, coal with a volatile content of between about 18 and 22% is used. If necessary, a suitable mixture of high-volatile coal (gas coal, fat coal) and low-volatile coal (lean coal, anthracite) can be fired.

Table 1: Comparison of various fuels

constituents and properties	unit	medium volatile coal	heavy fuel oil	low-sulphur fuel oil	natural gas (from Slochteren)
C	% by weight	88.4	86.0	85.0	57.9
H	% by weight	4.9	11.7	13.5	18.9
S	% by weight	1.2	1.5	0.5	
N	% by weight	1.3	0.2		21.8
O	% by weight	4.2	0.6		1.4
water (raw fuel)	% by weight	2–7	0.1–0.2	traces	
ash (raw fuel)	% by weight	6–20	bis 0.1	traces	
density at 0° C, 1013 mbar	kg/m^3				0.830
density at 15° C, 980 mbar	kg/m^3		930–950	830–860	
bulk density	kg/m^3	900–950			
calorific value H_u (net value for coal free from water and ash)	kJ/kg, kJ/m^3	34750	40200–41450	42700	31600[1]
gross calorific value	kJ/kg, kJ/m^3	35590	42700–44000	45550	35100[1]
theoretical flame temperature, without dissociation and air preheating	°C	2155	2120	2160	2010
minimum air requirements[1]	m^3/kg, m^3/m^3	9.04	10.76	11.13	8.33
minimum air requirements referred to H_u[1]	$m^3/10^3$ kJ	0.260	0.261	0.261	0.264
minimum combustion gas (waste gas, moist)[1]	m^3/kg, m^3/m^3	9.35	11.42	11.89	9.35
minimum combustion gas (waste gas) referred to H_u[1]	$m^3/10^3$ kJ	0.269	0.277	0.278	0.296
constituents of minimum combustion gas	% by volume	17.4	13.7	13.3	9.6
	% by volume	7.6	12.5	12.7	18.5
	% by volume	75.0	73.8	74.0	71.9

[1] Referred to standard conditions (0° and 1013 mbar)

The coal, or coal mixture, is dried and ground to a suitable fineness specified, in Germany, as a certain residue retained on the DIN 1171 test sieve with 0.09 mm aperture size: this residue is defined as half the content of volatile matter in the coal (expressed in per cent). See Fig. 1.

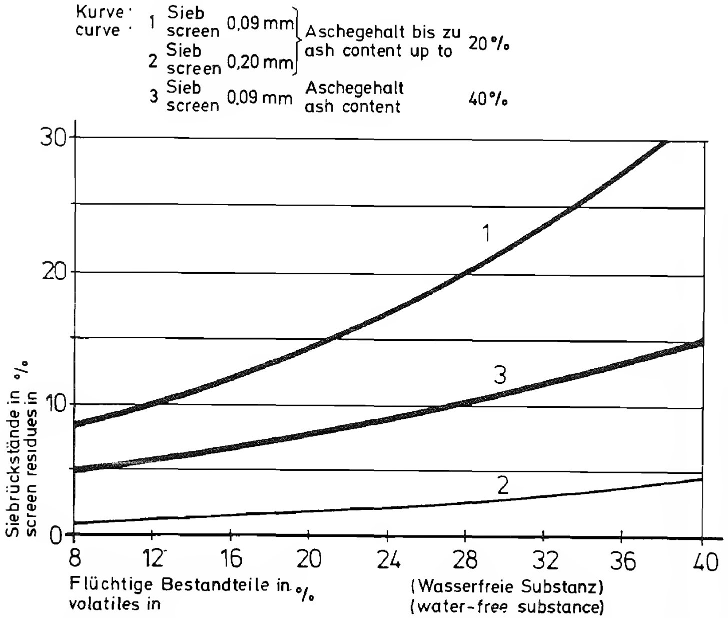


Fig. 1: Sieve residue of coal as a function of the volatile content and ash content (from KHD Humboldt Wedag AG, Cologne)

Lignite (brown coal) is another fuel fired in pulverized form. This substance generally has a volatile content in excess of 50%, necessitating extra care and precautions against fire and explosion hazard during grinding, storage and handling. Arrangements for flooding the system with an inert gas to suppress a possible outbreak of fire should be provided.

The safety regulations for dealing with pulverized fuels, issued by the Berufsgenossenschaft Steine und Erden (the employers' liability insurance association for the German pit and quarry industry) must be scrupulously complied with.

The ash from the coal becomes incorporated in the clinker in the course of the sintering process, a fact that must be duly taken into account in determining the

chemical composition of the raw meal. Coal with more than 20% ash content will in some cases necessitate the addition of pure (high-grade) limestone to the raw mix in order to compensate for this.

4.1.2 Oil

As a rule, rotary kilns are fired with "heavy fuel oil" (designated in Germany as "fuel oil S"). Its properties are listed in Table 1. The oil used should have as low a sulphur content as possible, or otherwise a low-sulphur oil ("fuel oil E1") should be used, though admittedly it is more expensive.

Oil is viscous at low temperatures and has to be heated to approximately 50° C for discharging it from tanks, pumping it and generally handling it. For good atomization in burners the oil temperature has to be further raised to 120° C. The heat transfer medium used for the purpose is mainly a special oil (thermal oil).

Fuel oil pumping pressures range from 40 to 60 bar, with flow velocities of about 0.2 m/sec on the suction and 0.4 m/sec on the delivery side of pumps as maximum values. To achieve optimum firing conditions the pressure and temperature of the oil fed to the burners should be as nearly constant as possible.

4.1.3 Gas

Gaseous fuel for cement kilns is predominantly natural gas, the properties of which are given in Table 1. It is supplied by pipeline at pressures ranging from 10 to 70 bar, which are reduced to between 3 and 10 bar in a pressure regulating station for use in the cement works.

4.2 Storage of fuels

4.2.1 Coal

Coal can be stored in outdoor stockpiles, in bunkers or in silos. It is usually supplied in the form of "washed smalls" and suitably large discharge cross-sections at bunker outlets, etc. should be provided in order to prevent choking, because the coal has poor flow properties. The requisite storage capacity will depend on local conditions and on the rate of fuel consumption of the works. Safety regulations applicable to coal storage must be duly complied with.

4.2.2 Oil

This fuel is stored in one or more tanks equipped with a tunnel accommodating the discharge pipe and with a heating system for suitably reducing the viscosity of the oil. Storage capacity will likewise depend on the particular conditions relating to the cement works concerned. Oil storage tanks and associated installations are subject to special safety regulations.

4.2.3 Gas

Normally, no storage tanks are required for natural gas, as this fuel is supplied at a constant rate by pipeline.

4.3 Preparation of fuels

4.3.1 Coal

Coal is generally supplied to the works in the form of "washed smalls" and has to be dried and ground before it can be fired in the kilns. The same requirements are applicable to lignite unless it is supplied ready for firing, i.e., in pulverized form, as is sometimes done.

4.3.2 Oil

Apart from having to be heated to 50° C for pumping and to 120° C for firing, as already stated, it requires no preparatory treatment. Oil filters should be provided, however.

4.3.3 Gas

Apart from a pressure reducing and regulating station, natural gas does not require any treatment at the cement works either.

4.4 Firing systems

4.4.1 Pulverized coal firing

A distinction can be drawn between direct and indirect firing systems for rotary kilns, with semi-direct firing as an intermediate solution.
With direct firing the coal is ground and dried (as a simultaneous operation) and then supplied direct, i.e., without intermediate storage, from the mill to the burner. All the exhaust air from the mill is fed as primary air to the kiln.
With the indirect system the coal is likewise simultaneously ground and dried, but is then stored in a bunker or bin.
Semi-indirect firing denotes a system whereby the primary air flow can be reduced to such an extent as is compatible with adequate removal of the moisture from the coal grinding/drying mill, while the rest of the mill exhaust air is returned to the mill.
Direct firing operates in conjunction with an additional primary air blower, enabling the pressure with which the air is blown into the kiln to be adjusted to the desired value of 120–150 mbar. With the indirect firing system all the exhaust air from the mill is dedusted and then discharged into the atmosphere (Figs. 2, 3 and 4).
To be sure of maintaining a constant flame in the kiln the fuel must be fed at a constant rate – in terms of weight of pulverized coal supplied per unit time.
The combustion air supplied to the kiln comprises primary air and secondary air, the former being the air which serves as the carrying medium for blowing the pulverized coal into the kiln. This air should preferably be preheated and its volumetric flow rate be kept as low as possible in order to achieve maximum utilization of the very hot secondary air, which arises as exhaust air discharged from the clinker cooler.

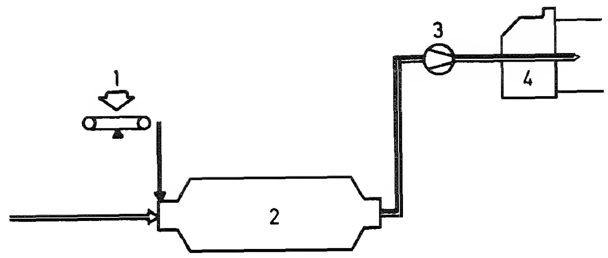


Fig. 2: Direct firing (from Dürr, 1979)

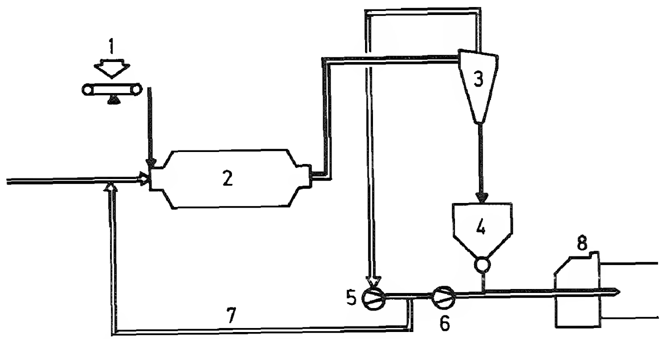


Fig. 3: Semi-direct firing (from Dürr, 1979)

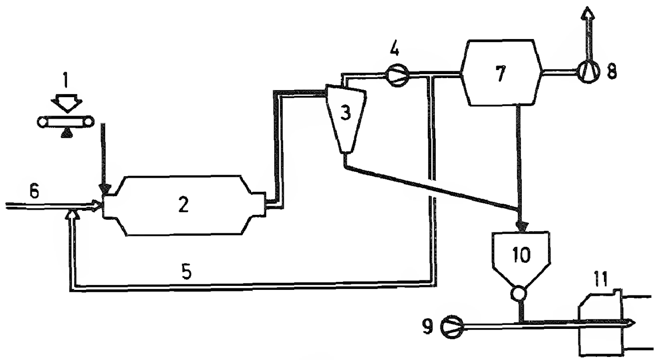


Fig. 4: Indirect firing (from Dürr, 1979)

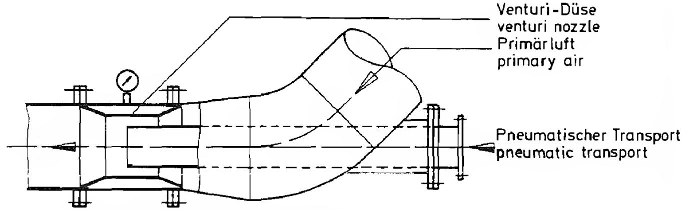
The minimum amount of air required for combustion will depend on the heat consumption of the kiln and can be approximately calculated from:

$$L_v = 0.261 \times \frac{\text{specific heat consumption (kJ/kg of clinker)}}{10^3}$$

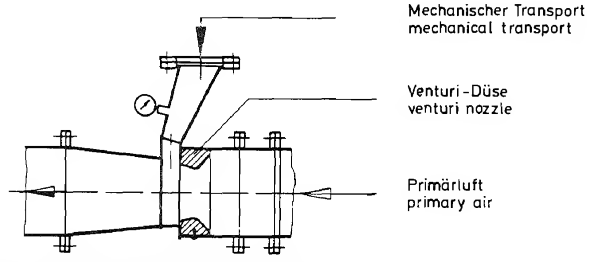
= Nm^3 of air/kg of clinker.
(Nm^3 denotes "standard cubic metre", i.e., at 0°C and 1013 mbar.)
Example: For a specific heat consumption of 3200 kJ/kg of clinker:

$L_v = 0.261 \times 3200/10^3 = 0.84\ Nm^3$/kg of clinker.

For satisfactory combustion a certain air excess (about 5 to 15%) over and above the minimum amount will be required.
The coal firing burner is essentially a plain tube provided with a nozzle-like outlet at its tip. The fuel feed arrangement is as shown in Fig. 5. The exit velocity of the air



Mischdüse für pneumatischen Kohletransport
mixing nozzle for pneumatic coal transport



Injektordüse für mechanischen Kohletransport
injector nozzle for mechanical coal transport

Fig. 5: Feed system for pulverized fuel (from KHD Humboldt Wedag AG, Cologne)

carrying the pulverized fuel should be between 40 and 80 m/sec. The primary air requirement is 0.7–1.8 Nm^3/kg of coal. High-volatile coal will normally require a lower primary air rate than low-volatile coal.

4.4.2 Oil firing

Various types of atomizing nozzle have been developed for achieving efficient atomization of the oil heated to about 120°C. Some of these are illustrated in Figs. 6 and 7. The oil is supplied to the burner nozzle at a pressure which can be

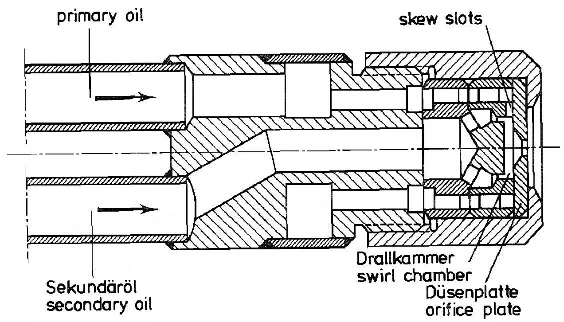


Fig. 6: Oil burner (from Pillard Feuerungen, Taunusstein)

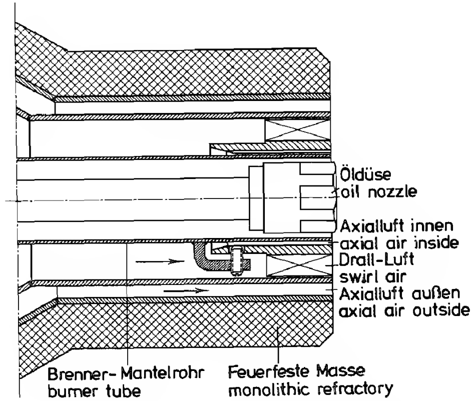


Fig. 7: Oil burner (from Pillard Feuerungen, Taunusstein)

controlled between about 3 and 10 bar. The length and width of the flame are determined by imparting a swirling motion to a certain proportion of the oil (primary oil) or of the primary air

4.4.3 Natural gas firing

Because of its relatively simple control and convenience of handling, natural gas has gained wide acceptance as a fuel in the cement industry. No primary air is needed, and the hot secondary air available as exhaust from the clinker cooler can be utilized to a considerable extent. Fig. 8 shows a commonly used type of gas burner, operating with exit velocities of up to 600 m/sec at gas pressures ranging up to about 4.5 mbar. This burner system can moreover be designed for firing oil or pulverized coal as a second or alternative fuel (Fig. 9).
As in an oil-fired kiln, the shape of the gas flame can be modified by varying the ratio of the axial to the swirling flow rate of the fuel A factor to be taken into account in determining the capacity of various parts of the equipment, especially the exit gas fan, is that with gas firing the quantities of exit gas are larger than with coal or oil.

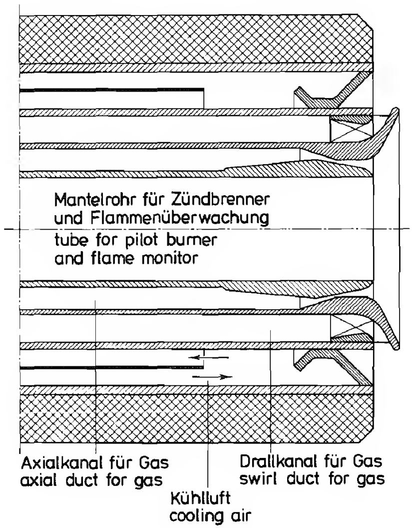
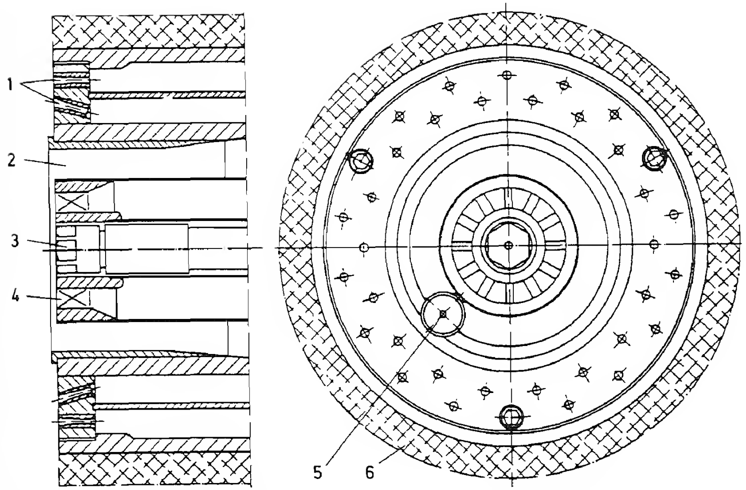


Fig. 8: Gas burner (from Pillard Feuerungen, Taunusstein)



1 Gasbrennerdüse. axial/radial
gas burner nozzle: axial/radial

2 Ringspalt für Kohlenstaub und Primärluft
annular gap for admitting pulv. coal a. air

3 Ölbrennerdüse
oil burner nozzle

4 Wirbelluftleitung für Öl und/oder Kohlenstaub
whirl airduct for oil and/or pulv. coal firing system

5 Zündbrenner
ignition burner

6 Feuerfeste Ummantelung
refractory bricklining

Fig. 9: Three-component burner with high-pressure gas burner (from KHD Humboldt Wedag AG, Cologne)

The natural gas burner is designed for a turndown ratio of 30:1, so that the kiln can, thanks to this range of firing control, be started up from cold without any need for auxiliary burners.
It should, finally, be noted that special combined firing burners have been developed for utilizing low-grade fuels. With the aid of a back-up flame (fed with gas or oil) such equipment can fire inferior grades of coal and certain combustible waste materials.

4.5 Residence time of the material and loading factor of the kiln

The progress of the feed material through the rotary kiln may be characterized by subcritical or supercritical motion. In the former case the material moves in an

oscillatory fashion, being raised some distance in contact with the rotating wall of the kiln and then sliding down again. Under such conditions there is hardly any mixing of the material, and heat transfer is correspondingly poor. This being so, supercritical motion is the what should be aimed at, i. e., all the material particles should proceed along circular rising paths and fall back onto the slope of the material in the kiln.
There are two components that determine the overall motion of the material through the kiln:

(a) The material moves in the direction of the longitudinal axis because of the slope of the kiln, which is generally between 3 and 4%, corresponding to angles of 1°43′ to 2°17′.
(b) It moves in a direction perpendicular to the kiln axis, this being due to the rotation. According to Heiligenstaedt (1951) the sliding angle ρ of the material in the kiln has the following values:

raw meal, warm, 0–0.2 mm size	35°
raw meal, matured, 0–0.2 mm size	45°
cement clinker, 0–50 mm size	35–40°

The resulting motion is characterized by an angle α which can be calculated as follows:

$$\sin\alpha = \frac{\sin v}{\sin\rho}.$$

The hourly throughput of a dry-process rotary kiln can be approximately calculated from.

$$L = \varphi\cdot\frac{\pi d^2}{4}\cdot d\cdot\pi\cdot\tan\alpha\cdot 60\cdot n,$$

or:

$$L = 148\cdot\varphi\cdot d^3\cdot\tan\alpha\cdot n.$$

In this formula the loading factor (or filling ratio) φ is defined as the ratio of the material-filled cross-sectional area F to the total internal cross-sectional area of the kiln f, i.e.,

$$\varphi = \frac{F}{f}.$$

The time of passage of the material through the kiln can be approximately calculated from.

$$\vartheta = \frac{l}{d}\cdot\frac{1}{\pi\cdot n\cdot\tan\alpha}.$$

The average velocity of the material in the kiln is:
$w = \pi\cdot d\ \ n\cdot\tan\alpha,$

where.

- α angle of material motion
- ϑ time of passage
- v angle of kiln slope
- ρ sliding angle of material
- φ loading factor of kiln (about 0.08 to 0.12)
- d internal diameter of kiln in m
- f internal cross-sectional area of kiln in m^2
- F cross-sectional area of material in m^2
- l length of kiln in m
- L kiln throughput in m^3/hour
- n speed of kiln in min^{-1} (r.p.m.)
- w material velocity in m/min.

4.6 Thermal calculations

4.6.1 Calorific value of fuel

For coal the constituents per kg are indicated as follows:

C kg of carbon	S kg of sulphur
H kg of hydrogen	W kg of water
O kg of oxygen	A kg of ash.

With this information it is possible to calculate the (net) calorific value of the coal by means of the following formula:

$$H_u = 33900\,C + 121400\,(H - 1/8\cdot O) + 10500\,S - 2500\,W\ \text{(kJ/kg)}.$$

For heavy fuel oil the following formula gives a fair general average value:

$$H_u = 41\,280 \pm 300\ \text{(kJ/kg)}.$$

For gaseous fuels the calorific value can be calculated from the volumetric percentages of the constituents (reckoned for standard conditions, i. e., at 0° C and 1013 mbar):

$$H_u = 126.33\ CO + 107.83\ H_2 + 358.83\ CH_4 + 643.45\ C_2H_6 + 932.07\ C_3H_8 + 1238.10\ C_4H_{10} + 595\ C_nH_m\ (kJ/Nm^3).$$

4.6.2 Calculation of exit gases

4.6.2.1 Oxygen requirement

The equation of the reaction representing the combustion of coal is.

$C + O_2 = CO_2$.
This corresponds to the quantitative balance:

1 kmol C + 1 kmol O_2 = 1 kmol CO_2

or:

$12\,kg\ C + 32\,kg\ O_2 = 44\,kg\ CO_2$.

This means that the complete combustion of C kg of carbon requires $\frac{8}{3}$ C kg of oxygen. Furthermore:

$$H_2 + \frac{1}{2} O_2 = H_2O,$$

or:

$2\,kg\ H_2 + 16\,kg\ O_2 = 18\,kg\ H_2O$.
Hence: the complete combustion of H kg of hydrogen requires 8 H kg of oxygen.

$S + O_2 = SO_2$,

or:

$32\,kg\ S + 32\,kg\ O_2 = 64\,kg\ SO_2$.

Hence: the complete combustion of S kg of sulphur requires S kg of oxygen. Taking account of the oxygen already contained in the coal, the quantity of oxygen required for the complete combustion of 1 kg of coal is theoretically:

$$\frac{8}{3} C + 8H - O + S \text{ (kg)}.$$

4.6.2.2 Air requirement

1 kg of air contains 0.23 kg of oxygen and 0.77 kg of nitrogen.
For standard conditions:

1 m^3 of air = 1.293 kg of air,

hence:

1 m^3 of air = 0.21 m^3 of oxygen + 0.79 m^3 of nitrogen.

Hence the air requirement per kg of coal is:

$$L_k = \frac{^8/_3\,C + 8H - O + S}{0.23} \text{ (kg/kg)}.$$

$L_k = 11.6\,C + 34.78\,H - 4.35\,O + 4.35\,S$.

The air requirement in Nm^3 (m^3 under standard conditions) per kg of coal is

$L_v = L_k/1.293 = 8.89\,C + 26.67\,H - 3.33\,O + 3.33\,S$.

4.6.2.3 Exit gas from combustion of coal

The stoichiometric combustion of 1 kg coal produces a quantity of exit gas at least equal to:

$G_k = 1 + L_k$ (kg/kg) (theoretical).

For complete combustion the purely stoichiometric quantity of air is insufficient, however. In practice a certain air excess has to be provided, usually expressed as an "air excess factor" n. The quantity of exit gas formed per kg of coal is then:

$G_k = 1 + n \cdot L_k$ (kg/kg) (combustion with excess air).

4.6.2.4 Exit gas from cement burning process

In cement burning the carbon dioxide and water vapour from the raw material are additional to the gases of combustion arising from the fuel.

Assumptions:

1 kg of raw meal contains x kg of water and y kg of $CaCO_3$; to produce 1 kg of cement clinker requires z kg of raw meal and k kg of coal

The exit gas per kg of clinker will be composed as follows:

from combustion of fuel	=	$(1 + n \cdot L_k) \cdot k$	kg
from raw meal (carbon dioxide)	=	$z \cdot y \cdot 0.44$	kg
from water content (vapour)	=	$z \cdot x$	kg

Total: $G_k' = (1 + n \cdot L_k) \cdot k + z \cdot (x + 0.44\,y)$ kg/kg of clinker.

1 kg of kiln exit gas corresponds to about 0.76 Nm^3

Hence $G_v' = 0.76\ [(1 + n \cdot L_k) \cdot k + z \cdot (x + 0.44 \cdot y)]$ Nm^3 of exit gas/kg of clinker.

4.6.2.5 Expansion of gases

An ideal gas undergoes an expansion of 1/273 of its volume per degree (°C) of rise in temperature. Hence the volume of the kiln exit gas (in m^3) at t°C is

$V_t = V_n\ (1 + t/273)$.

Where V_n is the volume in Nm^3 (i.e., at 0°C and 1013 mbar).

4.6.3 Heat consumption of clinker burning process

Assume that the kiln is fired with (dry) coal containing the following quantities of constituent materials per kg:

C = 0.78; H = 0.04; O = 0.10; S = 0.04; H_2O = 0.02; ash = 0.02.

The calorific value of this coal can be calculated from the formula in Section 4.6.1:

$H_u = 33900 \times 0.78 + 121400 \times 0.0275 + 10500 \times 0.04 - 2500 \times 0.02 = 30150$ kJ/kg.

For establishing the heat balance it is necessary to calculate the amounts of heat entering and leaving the kiln system (for a reference temperature of 20° C). The sensible heat can be calculated from the following formula:

$Q'_{sens} = m' \cdot c_p \cdot (t - 20)$ (kJ/kg of clinker).

where:

m' = mass flow referred to clinker, in kg/kg
t = temperature of the material entering or leaving the system, in °C
c_p = specific heat of the material, in kJ/kg · K.

For calculating the specific heat c_p (in kJ/kg · K) the following approximate equations are available:

for carbon dioxide:	$0.80 + 0.000461\,t$
for clinker:	$0.76 + 0.000297\,t$
for water vapour:	$1.76 + 0.000775\,t$
for exit gases:	$0.96 + 0.000209\,t$
for raw material:	$0.88 + 0.000293\,t$.

The fuel consumption of the kiln referred to clinker can be calculated as:

$$Q'_{fuel} = H_u \cdot k \text{ (kJ/kg of clinker).}$$

The heat of formation of clinker can be calculated by the method indicated by H. zur Strassen (1957) and presented in "Berechnungsunterlagen für Ofenversuche des VDZ", 1959. It is approximately 1600–1850 kJ/kg of clinker. For evaporating the free (uncombined) water in the raw meal or slurry the heat requirement is 2453 kJ per kg of water. For G'_{H_2O} kg of water per kg of clinker the heat required for evaporating this water is:

$Q'_{evap} = G_{H_2O} \times 2453$ (kJ/kg of clinker).

The heat intake from sensible heat of the fuel, cooling air and feed material, and from any combustible matter contained in the latter, is generally in the range of 1 to 3% of the total heat consumption and will be neglected in the following examples.

4.6.3.1 Long wet-process kiln

The following data will be adopted

water content in raw slurry = 36%
water content referred to raw meal = 0.56 kg/kg of meal, 1 kg of raw meal contains 0.76 kg of $CaCO_3$
external (ambient) temperature = 20° C
exit gas temperature = 200° C
temperature of clinker = 200° C
heat of clinker formation = 1830 kJ/kg
fuel consumption = 20% = 0.2 kg of coal/kg of clinker assuming a calorific value of 30150 kJ/kg of coal, i.e., heat input = 6030 kJ/kg of clinker
air excess factor n = 1.2.

Quantity of raw meal required to produce 1 kg of clinker is 1.54 kg (see Labahn/Kaminsky, 1974, p. 65)

Quantity of **exit gas** produced:

from fuel: $0.2\,[1 + 1.2\,(11.6 \times 0.78 + 34.78 \times 0.04 - 4.35 \times 0.10 + 4.35 \times 0.04)]$		= 2.65 kg/kg
from water: 0.56×1.54		= 0.86 kg/kg
from raw meal (CO_2): $0.44 \times 0.76 \times 1.54$		= 0.52 kg/kg
kiln exit gases per kg of clinker:	total	= 4.03 kg/kg
corresponding volume of gas under standard conditions		= 3.06 Nm^3/kg

Heat balance in kJ per kg of clinker:

heat of clinker formation	= 1830
exit gas heat loss from fuel: $2.65\,(0.96 + 0.000209 \times 200) \cdot (200 - 20)$	= 478
exit gas heat loss from raw meal carbon dioxide: $0.52\,(0.80 + 0.000461 \times 200) \cdot (200 - 20)$	= 84
exit gas heat loss from water vapour: $0.86\,(1.76 + 0.000775 \times 200) \cdot (200 - 20)$	= 296
water evaporation: 0.86×2453	= 2110
waste heat in clinker: $1.0\,(0.76 + 0.000297 \times 200) \cdot (200 - 20)$	= 147
radiation losses, etc. (residual value)	= 1085
calculated heat requirement per kg of clinker	= 6030 kJ

4.6.3.2 Rotary kiln with cyclone preheater and exit gas utilization in a roller mill

Rotary kilns with cyclone preheater equipment generally discharge exit gas at a temperature of between 320° and 360° C. This gas contains only about 12% moisture and is therefore very suitable for the drying of raw materials. In the following example a rotary kiln with cyclone preheater operating in combination with grinding/drying mill, in which the exit gas heat is utilized, will be considered.

The assumed data are:

exit gas temperature = 360° C
temperature of clinker = 150° C
water content of raw meal = 0.5%
water quantity = 0.005 kg/kg of dry raw meal
fuel consumption = 10.6% = 0.106 kg of coal/kg of clinker assuming a calorific value of 30150 kJ/kg of coal, i.e., heat input = 3190 kJ/kg of clinker
clinker output of kiln = 3000 t/day
exit gas temperature on discharge from roller mill = 120° C

$$\text{mill throughput} = \frac{3000 \times 1.6}{24} = 200\text{ t of raw meal/hour}$$

moisture content of raw material = 7%
residual moisture in (dried) raw meal = 0.5%
For other data see Section 6.3.1 (long wet-process kiln).

Quantity of **exit gas** produced:

from fuel: 0.104 [1 + 1.2 (11.6 × 0.78 + 34.78 × 0.04 − 4.35 × 0.10 + 4.35 × 0.04)]	= 1.40 kg/kg
from water: 0.005 × 1.54	= 0.01 kg/kg
from raw meal (CO_2): 0.44 × 0.76 × 1.54	= 0.52 kg/kg
kiln exit gases per kg of clinker: total	= 1.93 kg/kg
corresponding volume of gas under standard conditions	= 1.47 Nm^3/kg

Heat balance in kJ per kg of clinker:

heat of clinker formation	= 1830
exit gas heat loss from fuel: 1.40 (0.96 + 0.000209 × 360) · (360 − 20)	= 492
exit gas heat loss from raw meal carbon dioxide. 0.52 (0.80 + 0.000461 × 360) (360 − 20)	= 171
exit gas heat loss from water vapour: 0.01 (1.76 + 0.000775 × 360) · (360 − 20)	= 7
water evaporation: 0.01 × 2453	= 25
waste heat in clinker. 1.0 (0.76 + 0.000297 × 150) · (150 − 20)	= 105
radiation losses (residual value)	= 560
calculated heat requirement per kg of clinker	= 3190 kJ

Exit gas utilization

The following exit gas flow is available for drying the raw material·

$$\frac{3000 \times 10^3 \times 1.47}{24} = 183750\ Nm^3/\text{hour, including false air about } 200000\ Nm^3/h.$$

For producing 200 t of raw meal per hour the quantity of water to be evaporated (Labahn/Kaminsky, 1974, pp. 118 and 125) is·

$$W_a = T_r \cdot \frac{w - w_r}{100 - w} = 200 \times 10^3 \times \frac{7 - 0.5}{100 - 7} = 13980 \text{ kg of water/hour}$$

For a gas temperature of 360° C on entering and 120° C on leaving the mill, the following quantity of exit gas is needed:

$$G_h = \frac{W_a \cdot k}{t \cdot s_g} = \frac{13980 \times 5400}{240 \times 1.36} = 231300\ Nm^3/\text{hour}$$

According to Labahn/Kaminsky (1974, page 122), water evaporation requires a heat input of 5400 kJ/kg. Hence the requirement for evaporating 13980 kg of water per hour is

13980 × 5400 = 75.5 × 10^6 kJ/hour.

The available exit gas quantity is 200000 Nm^3/hour at a temperature of 360° C. After utilization of the heat in this gas, its temperature on discharge from the mill is 120° C. Hence the heat derived from the gas is:

200000 × (1.36 × 360 − 1.30 × 120) = 66.7 × 10^6 kJ/hour.

In this example the exit gas does not provide sufficient heat for drying the raw material. 8.8 × 10^6 kJ/h will have be supplied to the mill by extra heat. This heat can be obtained from a separate air heater or, alternatively, may be available as waste heat in the exhaust air from a grate cooler (see below)
The heat utilized from the exit gas by raw material drying during grinding is.

$$\frac{66.7 \times 10^6 \times 24}{3000 \times 10^3} = 534 \text{ kJ per kg of clinker.}$$

If this heat is deducted from the heat consumption of the kiln, the latter is reduced from 3190 kJ to 2656 kJ per kg of clinker.
A further lowering of the heat consumption of the rotary kiln can — under appropriate operating conditions — be achieved by utilization of the heat contained in the exhaust air from the clinker cooler (grate cooler) in so far as this air cannot be supplied as secondary or tertiary air to the kiln system. An additional heat recovery of about 260 kJ per kg of clinker can thus be gained from the exhaust air of the cooler.
For determining the capacity of the dust collection equipment, it will be necessary to determine the exhaust gas discharged from the grinding/drying plant:

exhaust gas from rotary kiln	= 183800 Nm^3/hour
water vapour from drying: 13980 × 1.244	= 17390 Nm^3/hour
total	= 201190 Nm^3/hour.

At 120° C exit temperature of the gas discharged from the mill this is equivalent to:

$$\frac{201190 \times 393}{273} = 289625\ m^3/\text{hour}$$

Allowing 20% (= 57925 m^3/hour) to take account of infiltrated air, the total gas flow (at 120° C) to be treated by the dust collection equipment will be about 347550 m^3/hour
Similar calculations are valid for all types of kiln. For the dry-process rotary kiln with preheater the heat losses through the walls amount to about 23% of the total

heat consumption. The corresponding figures for the wet-process rotary kiln and the shaft kiln are about 17% and 12% respectively.

Approximate determination of **specific kiln output:**

The specific output of the kilns can be used as a basis for comparing different clinker burning processes. It is a quantity obtained from the clinker output in t/day and the internal volume of the kiln in m^3, the latter being calculated from the internal diameter d and length L of the kiln:

$$V_{int} = \frac{1}{4}\pi \cdot d^2 \cdot L = \frac{1}{4}\pi\ (D - 0.4)^2 \cdot L.$$

In this formula d has been taken as approximately equal to the internal diameter within the shell D less 0.4 m to allow for the lining.

The following are some guiding values for specific kiln output:

	t/day m^3
long wet-process kiln	0.45 – 0.8
long dry-process kiln	0.5 – 0.8
kiln with cyclone preheater	1.5 – 2.2
kiln with grate preheater	1.5 – 2.2
kiln with precalcining	3.3
kiln with precalcining and tertiary air duct	3.5 – 5.0.

With the aid of these values it is possible to calculate approximately the kiln dimensions for achieving a specified clinker production rate, if the length/diameter ratio of the kiln is known, for which the following approximate values may be adopted:

long wet or dry kilns	L/D = 34
short kilns with preheater equipment	L/D = 16.

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5 Refractory linings

By D. Opitz

5.1 General

The function of refractory materials is to protect metal parts from coming into direct contact with flames or with very hot gases or solids. For example, boiler plate undergoes a marked decline in strength at temperatures above 400° C, while clinker temperatures are in the range of about 1350°–1550° C and the flames in kilns attain almost 1900° C.
The heat loss through the wall of the kiln must moreover be kept within acceptable limits. Even so, depending on the kiln system, between 12 and 22% of the heat evolved from combustion of the fuel is lost in this way.
About 10% of the heat given off by the flame and its associated combustion gases is first transmitted to the surface of the refractory lining from which in turn it is transferred to the feed material being processed in the kiln.
The rough surface of the refractory brickwork moreover promotes the supercritical motion of the material, so that effective mixing takes place and heat transmission from the hot gas to the material is improved.
Damage to the lining is liable to cause trouble which may necessitate shutdown of the kiln plant for repairs.
The cost of the lining amounts to between 1 and 2% of the cost of construction if the rate of consumption of refractory material is about 0.5–1.5 kg per tonne of portland cement clinker produced. About 35% of the lining in a modern kiln typically consists of dolomite brick, 35% of magnesite-chrome brick, and the remainder (30%) of fireclay brick, lightweight refractory brick, special brick and monolithic refractories.
Although large rotary kilns may have lower specific refractory consumption rates (i.e., per tonne of clinker), the risk of shutdown due to unexpected lining damage is much higher in such kilns than in smaller ones with shell diameters of not more than about 4 m. The introduction of "secondary firing" in the preheater, i.e., various forms of (pre)calcining, was associated with the advantage that the thermal rating of the burning zone – and therefore the severity of the conditions to

which the lining in this zone was exposed – was substantially reduced, especially in large kilns, thus resulting in a significant increase in the service life of the lining.
The "state of the art" in respect of refractory linings for cement kilns was reviewed in a literature analysis by Routschka and Majdic (1974), the results of which have been used by the present author. Subsequent research on the subject (1974–1978) will also be considered here, as far as possible.
In its "catalogue of refractory materials for cement works" (Merkblatt WE 10), 1968, the German Cement Works' Association (VDZ) lists a considerable number of refractories on the basis of information supplied by their manufacturers, together with guidance on the appropriate use of these materials. In principle, little has since changed in the range of refractories employed in cement manufacturing plant. Table 1 lists categories of materials, with indications on their application in the respective parts of kiln systems.
The various kiln zones may suitably be lined in accordance with the suggestions in Table 2, which distinguishes between kilns of 4.4 m diameter and larger ones.
When lined for the first time, the kiln hood is given a working (or inner) lining of high-alumina brick. For repairs to partly worn brickwork, provided that it is suitably stable and is properly cleaned, a new wearing layer can be formed with a refractory mixture applied with a spray gun.
The preheater, clinker cooler, exit gas ducts and other hot parts of the kiln system are lined with fireclay brick or fireclay concrete. Where high temperatures occur it may be necessary to use refractories with an alumina content above 44%.
Refractory bricks are tested for certain properties and/or uniformity of the consignment supplied. Guidance for the selection of refractories with reference to the mechanical, physical and chemical conditions of service is given by Majdic (1974). The criteria for selecting and using high-alumina and fireclay bricks in rotary cement kilns have more particularly been dealt with by Bartha (1978). The mechanical properties of magnesite bricks in relation to their composition and texture are described by Kienow/Jeschke/Doas (1977). Dolomite bricks are dealt with by Münchberg/Opitz/Stradtmann (1977).

5.2 Testing the properties

Cold crushing strength

If it is deficient, bricks are liable to suffer damage during handling and transport. As a rule, a strength of at least 15 N/mm^2 is required from this point of view. It is only very tentatively possible to draw inferences from the cold strength as to the strength and abrasion resistance of the brick at high temperatures. Such inferences are indeed not permissible at all in the case of brick containing glassy solidified melt. High cold strength may moreover be an indication of brittleness, associated with a tendency for the brick to fracture when subjected to flexural loading when incorporated in brickwork.

Table 1: Composition and properties of refractory bricks (approximate values for guidance) (from Merkblatt WE 10, VDZ)

where used	chemical composition % Al_2O_3	chemical composition % Fe_2O_3	refractoriness/ Seger cone	refractoriness under load t_a °C	cold crushing strength N/mm^2	true porosity %	bulk density t/m^3	remarks
chimney flue	30–33	2.5	31	1300	25	25	2.0	below 300°C
	26–29	2.5	28	1300	50	16	2.1	possibly acid-resistant
feed-end ring	30–33	2.5	31	1300	25	25	2.0	fireclay brick
	26–29	2.5	28	1300	50	16	2.1	acid fireclay brick
preheating zone	33–40	2.5	32	1350	25	25	2.0	fireclay brick
	–	–	–	–	12	45	1.5	lightweight refractory brick
calcining zone	40–42	2.5	33	1400	25	25	2.0	fireclay brick
	50	2.0	35	1500	50	19	2.3	alumina brick
transition zone	MgO: 65	9	–	1600	30	22	3.0	magnesite-chrome brick
	70	1.5	38	1600	50	18	2.6	high-alumina brick
burning zone	MgO + CaO: 96	0.8	–	1700	40	19	2.8	dolomite brick
	MgO: 85	9	–	1600	50	19	3.0	magnesite-chrome brick
cooling zone	50	2.0	35	1500	50	19	2.3	alumina brick
nose ring	50	2.0	35	1500	50	19	2.3	alumina brick
	SiC: 60	–	37	1500	50	22	2.3	silicon carbide brick
kiln hood	50	2.0	35	1500	50	19	2.3	alumina brick
	70	1.5	38	1600	50	18	2.6	high-alumina brick
clinker cooler	26–29	2.5	28	1300	50	16	2.1	acid fireclay brick
pellet and meal preheater	26–29	2.5	28	1300	50	16	2.1	acid fireclay brick
	50	2.5	35	1500	50	19	2.3	alumina brick
feed-end chamber firing	30–33	2.5	31	1300	25	25	2.0	fireclay brick
	40–42	2.5	33	1400	25	25	2.0	(for hottest parts)

Table 2: Examples of refractory linings for rotary kilns, from Merkblatt WE 10, VDZ

	kilns of 4.4 m diameter	kilns of more than 4.4 m diameter
nose ring	silicon carbide brick with 60% SiC or high-alumina brick with about 70% Al_2O_3	
cooling zone	2 m – 4 m high-alumina brick	2 m – 4 m magnesite-chrome brick
burning zone	$4 \times D_B$[1] dolomite brick[2]	$5 \times D_B$[1] dolomite brick[2]
transition zone	$2 \times D_B$ alumina or high-alumina brick	$5 \times D_B$ magnesite-chrome brick
preheating zone	fireclay brick with Al_2O_3 content decreasing towards feed end, or lightweight brick	

[1]) Kiln shell diameter in m

[2]) In kilns in which no coating is formed, or under very difficult burning conditions, magnesite or magnesite-chrome brick is used

Abrasion resistance

With regard to the cold abrasion resistance of refractory brick the comments made on the subject of cold crushing strength are valid here too. In a kiln with a stable coating attached to the refractory lining the abrasive action upon the latter is slight. Under such conditions there will be no abrasion of the lining by clinker at all except when initially heating up the kiln, i. e., before the lining has picked up any coating, or when an area of coating falls off, leaving the lining exposed until a fresh coating has formed.

Refractoriness

Refractoriness is defined as the property that allows the material to withstand high temperature. Although in this sense it is fundamental to the whole concept of a refractory product, it does not in itself offer much guidance on the practical usefulness thereof, except for comparison with other materials within the same category.

Refractoriness under load

The strain behaviour under load is measured; these tests provide an indication of the permissible thermal rating in the kiln.

Spalling resistance

The testing procedure for determining the spalling resistance (also known as thermal shock resistance) involves heating the brick to 950° C and then quenching it in water or, in the case of basic brick (which would undergo hydration), in air. The test results show considerable scatter. They should be used only for comparison within the same category of materials.

Chemical resistance

Contact reactions can be ascertained by heating relevant substances, e. g., kiln dust or cement clinker, in a crucible made of the refractory to be tested. Another type of test consists in heating appropriate cylindrical specimens placed one upon another. The results of such laboratory tests must be interpreted with caution. In actual practice, when the refractory is in service in the kiln, the continuous infiltration of gaseous and liquid substances into the brick gives rise to chemical conditions of a different character and different degree of severity.

Thermal expansion

The increase in dimensions of a refractory material caused by a rise in temperature is important with regard to the design of expansion joints in refractory linings.

Volume stability

The determination of this property consists in maintaining the brick at high temperature (without load) for some considerable time, then allowing it to cool, and finally measuring its dimensions, which are compared with the original dimensions of brick before the test. These test results are important only if the brick is to be used at temperatures which are higher than the firing temperature applied in its manufacture.

Thermal conductivity

This property can be significant in connection with heat losses through the refractory lining. Its determination is not entirely straightforward, however, and different measuring methods yield different results. Manufacturers' data should be accepted with caution.

Porosity

The apparent porosity, which is the ratio of the volume of the open pores to the bulk volume (expressed as a percentage), is of importance in that it gives an indication of the resistance of the refractory to attack by liquid and gaseous phases. The sealed pores – which together with the open pores constitute the true porosity – are not very important from the viewpoint of performance of the material.

5.3 Brick sizes

The problem of the most appropriate or efficient bricks sizes for the construction of rotary kiln linings has long been a subject of debate. In connection with the development of bigger and bigger kilns and the preference for "dry" bricking, special large bricks have been introduced as an alternative to the "VDZ" standard sizes used in Germany (see Majdic, 1974). Two series of standard sizes are commonly used (see Table 3), those of the "B" series being confined only to basic bricks. Providing each brick with a groove on its hot face as a means of conveniently checking that the bricks have been correctly installed has been introduced for dolomite bricks. Distinctive marking of bricks by means of several notches is being tried out for other types. Attempts have also been made to introduce tongue-and-groove bricks in order to obtain interlocking and thus improve the stability of the brickwork.

The lining in a rotary kiln is generally 200 mm thick, but can be varied to some extent according to kiln diameter:

D_B (m):	< 3.0	3.0–4.0	4.0–6.0	> 6.0
lining thickness (mm):	160	180	200	220–250

In the English-speaking countries the 280 mm high brick has additionally been introduced. These higher bricks are expected to fit more snugly in the ring and thus make for better lining stability. On the other hand, as experience shows, the thickness of the layers spalled off larger bricks is initially greater.

Table 3: Standard refractory brick sizes for rotary cement kilns (from VDI Code)

symbol	dimensions in mm				volume dm^3
	a	b	h	l	
216	103	86	160	198	2.99
316		92			3.09
416		94.5			3.13
516		96.5			3.16
716		98.3			3.18
16	85	80			2.61
218	103	84	180	198	3.33
318		90.5			3.45
418		93.5			3.50
518		95.5			3.54
618		97			3.56
718		97.5			3.57
18	85	80			2.94

symbol	dimensions in mm				volume dm^3
	a	b	h	l	
220	103	82	200	198	3.66
320		89			3.80
420		92.5			3.87
520		94.7			3.91
620		96.2			3.94
720		97			3.96
820		97.8			3.98
20	85	80			3.27
322	103	88	220	198	4.15
422		91.5			4.24
522		94			4.29
622		95.5			4.32
722		96.5			4.35
822		97.3			4.36
22	85	80			3.59
425	103	90	250	198	4.78
525		92.7			4.84
625		94.5			4.89
725		95.5			4.91
825		96.5			4.94
25	85	80			4.08
B 216	78	65	160	198	2.27
B 416	75	68			
B 16	85	80			2.61
B 218	78	65	180	198	2.55
B 318	76.5	66.5			
B 418	75	68			
B 518	74.5	68.5			
B 618	74	69			
18	85	80			2.94

standard size with average width of tapered face in mm

Table 3: Standard refractory brick sizes for rotary cement kilns (from VDI Code)

symbol	dimensions in mm a	b	h	l	volume dm^3
B 220	78	65	200	198	2.83
B 320	76.5	66.5			
B 420	75	68			
B 520	74.5	68.5			
B 620	74	69			
20	85	80			3.27
B 222	78	65	220	198	3.11
B 322	76.5	66.5			
B 422	75	68			
B 522	74.5	68.5			
B 622	74	69			
22	85	80			3.59
B 325	78	65	250	198	3.54
B 425	76.5	66.5			
B 525	75	68			
B 625	74.5	68.5			
B 725	74	69			
25	85	80			4.08

standard size with average width of tapered face in mm

5.4 Lining construction and demolition

A distinction is drawn between radial joints between the bricks in each ring and ring joints (or axial joints) between the successive rings.
Joints are weak spots in refractory brickwork, as regards both the mechanical strength of the lining structure and the penetration of liquid and gaseous (vaporized) substances into the lining. However, under circumstances where the thermal expansion of the lining (which generally exceeds that of the steel shell of the kiln) has to be compensated, such weak spots providing a certain amount of "give" are desirable and will, where possible, be methodically planned into the lining structure as expansion joints. The following are some approximate values for the thermal expansion and the operating temperature of the kiln lining.

	boiler plate	fireclay brick	magnesite brick	dolomite brick
linear thermal expansion (mm/m K)	0.011	0.005	0.011 – 0.022	0.014
temperature (°C)	150 – 450	600 – 800	1000 – 1200	1000 – 1200

The uncertainty associated with calculating the probable thermal expansion of the lining is due mainly to uncertainty about what temperature to take into account. As an approximate rule of thumb it is sometimes assumed that one-third of the lining's expansion is absorbed by the expansion of the kiln shell, one-third is absorbed by the ring joints and one-third by the expansion joints. For the design and construction of the expansion joints the brick manufacturer's recommendations should be complied with.
According to Konopicky (1957), the expansion joints should, as a rule of thumb, be designed to absorb half the thermal expansion that develops up to the service temperature of the refractory lining. Approximate values to be adopted are 0.5% for fireclay and silicon carbide brick, and 1.2% for magnesite-chrome and dolomite brick. The expansion joints are filled with cardboard, which burns away when the kiln is heated, leaving the joints clear. In general, it is more favourable to provide a larger number of narrower expansion joints than a smaller number of wider ones.
It may occur during a campaign (the working life of the lining between major repairs) that, as a result of the pores in the refractory becoming filled with sublimated vapours or penetrated melt (liquid phase), the coefficient of thermal expansion increases as compared with that of the newly installed (unused) brick. Otherwise the thermal expansion is reversible, so that, when the brickwork cools, joints open out again – though these will not necessarily coincide with the expansion joints originally formed.
If the expansion joints are made too wide, there will be the risk that bricks will drop out of the lining, whereas inadequately dimensioned expansion joints may give rise to excessive stresses in the brickwork which may result in premature destruction thereof or indeed cause trouble affecting the kiln structure itself
The normal brickwork joints used to be always constructed with mortar, but time-saving "dry" bricking is now increasingly used, especially for basic brick. Steel plates inserted into the joints between basic bricks undergo oxidation (ferrite formation) when the lining is heated and thus help to bond the individual bricks firmly together. Positive bonding during construction is obtained by sticking certain bricks to the kiln shell by means of a suitable adhesive, usually an epoxy resin glue. Possibilities of mortarless lining construction for rotary kilns are described by Zachwy/König/Eisemann (1975)
To enable new bricks to be introduced into the kiln and old sections of lining to be broken out and the debris removed, the kiln hood should be so designed that loading machines similar to those used in tunnel construction have proper access to the interior of the kiln.
As a rule, to achieve better stability of the rings of brick which form circular arches, the bricks employed are of suitably tapered shape. Various methods of supporting

the brickwork during lining construction are commonly used: glueing certain rows of bricks (strategically disposed around the circumference) to the kiln shell, using metal or wooden "centres" (as in conventional brick arch construction), using props of various kinds or screw jacks and timbers; fixing steel suppoórting ribs (rolled steel sections) to the inside of the shell by bolting them to nuts welded to the shell.
The linings in large rotary kilns are usually installed by means of the "glue" method or the "centre" method. With the latter, a spreader jack is inserted into the final gap before closing each brickwork ring in order to compress the bricks in the ring and thus ensure that they are sufficiently tight. Properties of adhesives for the glueing of refractory bricks have been investigated by Steinbiss (1975).
Like the bricking operations, the demolition of the brick lining has to be performed with special equipment, more particularly in large kilns. Mobile machines are used for transporting the demolished material. Manholes provided in the kiln shell reduce the transport distances.

5.5 Drying, heating-up, shutdowns

When the refractory lining has been installed for the first time and also after subsequent repairs, more particularly with mortar joints, it is necessary to dry the lining before heating-up. The lining in the preheater system of a kiln should be heated for ten days, after which a heating period of three days will suffice for the rotary kiln itself. After repairs to the lining, a heating-up period of between 12 and 36 hours will generally suffice, but care must be taken not to raise the temperature too rapidly, as this may cause considerable thermal stresses in the brickwork. Recent research (Erni/Saxer/Schneider, 1979) has highlighted the danger of constriction of the kiln shell under a tyre if the kiln is heated at so fast a rate that there is a significant time lag in the temperature rise of the tyre as compared with that of the kiln shell. With the usual tyre clearance of 9 mm (cold) with about 30 mm relative movement, harmful constriction of the shell by the colder tyre is liable to occur if the temperature of the tyre is more than 150° C lower than that of the shell. Small kilns can be heated up more quickly after repairs than big ones. As an approximate guide for larger kilns. allow 1 hour more heating-up time for each 100 t/day capacity above 1500 t/day, starting from a basic time of not less than 12–36 hours.

5.6 Thermal insulation

With the present high cost of energy it is imperative to keep heat consumption to a minimum.
To obtain better thermal insulation, the refractory lining of the preheating zone, may be constructed in two layers, namely, a dense and strong temperature-resistant working lining and a backing consisting of a porous grade of refractory brick. This "back-up insulation", usually between 40 and 80 mm thick, can be installed behind a basic working lining of dense fireclay brick, lightweight refractory brick or insulating brick, the characteristic feature of any such composite

lining being that the strength decreases and the heat insulating capacity increases from the hot face to the cold face. The extra cost of two-layer lining construction can be reduced by using two-layer bricks, i. e., consisting of lightweight refractory bonded to dense fireclay forming the working lining, these two layers respectively corresponding to one-third and two-thirds of the overall thickness of the composite brick. Strong interconnection of the two layers is aided by their indented interlocking. Even better insulation is provided by lightweight refractory bricks of adequate strength as the working lining of the preheating zone. Particularly the high-silica grades of brick form a glazed layer at the operating temperatures, which inhibits attack by lime or volatilized alkalis.
In the burning zone (clinkering zone) the simplest and cheapest form of thermal insulation is provided by a firmly adhering and sufficiently thick coating on the hot brickwork face. But if the conditions in the kiln are such that little or no coating is picked up by the lining, the desired reduction in heat loss can be obtained by installing back-up insulation. On the other hand, in a kiln in which the conditions for good coating are right, such an insulation would reduce the ability of the lining to take on a good coating, and for this reason it is a counter-productive measure in the burning zones of most rotary cement kilns.
Tiles and slabs incorporating ceramic fibres offer the advantage of reduced thickness in conjunction with equally good heat-insulating capacity, but they suffer from the drawback of limited strength, so that they can suitably be installed only in relatively small kilns (up to about 3.6 m diameter) and in the static parts of other kiln plants.
Thermal insulation problems relating to cement kilns were dealt with in considerable detail at the 17th International Refractories Colloquium on the subject of "refractory materials in the cement industry", held at Aachen in 1975. The papers presented on that occasion have been published in No. 5/1975 of the journal "Berichte der deutschen Keramischen Gesellschaft".

5.7 Lining wear

Basic linings in the burning zones of cement kilns undergo wear mainly as a result of spalling of relatively thin layers (e. g., 60–80 mm in thickness) of the brickwork which have undergone considerable chemical and mineralogical changes in service.
A reaction zone characterized by the formation mainly of C_2S is formed at the interface of the magnesite brick material and the clinker coating. As a result of accretive crystallization the working face of the brick becomes brittle and also its texture becomes denser. Alternatively, the brick may be subjected to severe attack by liquid (clinker melt), destroying the bonding phase of the refractory material and leaving a porous surface layer behind. Within the brick itself a zonal structure develops as a result of migration of the liquid phases originally present in the brick towards the cold face and of subsequent penetration of clinker melt into the brick.
Alkali sulphate and alkali chloride may form coatings on the refractory lining in the preheater and in the feed end region of the rotary kiln. If high temperatures prevail

in the kiln, these deposits will penetrate into the lining and fill up the cracks and joints, and also the pores within the brick, to a certain depth from the hot face. In the absence of an effective coating in the burning zone and with high process temperatures, the alkali melts may completely penetrate the lining and form a deposit – a crystallized layer – between it and the steel shell.

The zonal and structural changes that the brick undergoes during its service in the kiln may give rise to loosening (loss of cohesion) of the material structure and loss of mechanical strength. Besides, thermal expansion may increase. The hot flexural strength of the brick may be seriously impaired during the campaign.

Alumino-silicate refractories are destroyed by contact with cement clinker in consequence of reactions involving melt formation. The layer of liquid formed at the hot face of the lining in this way prevents the attachment of a permanent coating, so that the lining wears off fairly rapidly in thin layers. A grade of brick with a higher alumina content can withstand somewhat higher temperatures without suffering premature wear. Alumina-based refractories are not used in the burning zones of large kilns, however, because here even a very high alumina content will not protect the brick from melt formation at sintering temperature.

In general, factors which increase the rate of refractory brick wear are·

– frequent plant operating interruptions;
– non-uniform composition of raw material;
– high alkali content in raw material,
– low sintering tendency of the material;
– unstable kiln shell (excessive cross-sectional distortion),
– unstable flame,
– variable coating and ring formation.

Coating and ring formation directly and indirectly affect the service life of the lining. A relatively thin coating (up to 0.2 m) is beneficial because it increases the life of the brickwork it covers. But when objectionable thicker coatings are dislodged, the lining is subjected to very severe loads which are likely to shorten its useful life.

5.8 Coating and ring formation

The higher the proportion of liquid (clinker melt) formed by the feed material in the burning zone, the more readily will the refractory lining pick up a coating. In a coal-fird kiln ash from the coal will encourage coating formation. In especially unfavourable cases, however, the ash may become concentrated in the upper part of the burning zone and form a so-called ash ring there.

Coating will form more easily in the preheating and in the calcining zone according as the feed material contains more alkali salt melt. Secondary constituents of the fuel, such as compounds of sulphur and chlorine, promote such coating formation.

In both these cases, kiln feed material and dust adhere to the lining by the adhesive action of the melt. For equal melt content the adhesion strength depends on the temperature. This strength is highest in the 1200°–1240°C range in the burning zone. For this reason very thick coating rings may form at the two ends of this zone.

These formations may prove troublesome to kiln operation and have to be removed. If the flame is very short, sticking of the clinker to the inlet chute of the clinker cooler may occur In the preheating zone the strength of the coating consisting of salt melt is highest at temperatures of 900°–1000°C. In this temperature range several rings will form or, in short kilns with preheater equipment, thick coatings in the hot part of the latter. Depending on the chloride content in the sticky melt, the temperature at which the coating attains its highest strength may be below 900°C and even as low as about 600°C, however.

In the high-temperature parts of the kiln the tendency to objectionable coating ring formation is greater with raw meal of variable composition than with raw meal whose composition remains reasonably constant. In the preheating zone of the kiln, cyclic phenomena involving repeated volatilization and condensation of alkali salts, resulting in excessive concentrations of these substances in the feed material undergoing processing, promote ring formation. Discharging some of the alkali-laden gas from the kiln inlet through a so-called bypass (thus avoiding the preheater) provides a means of keeping the cyclic build-up in concentration of volatile substances within acceptable limits.

Besides salt melts, double compounds such as spurrite composed of calcium silicate and calcium carbonate or sulphate spurrite composed of calcium silicate and calcium sulphate may give rise to "dry" coating formation, the deposit being consolidated by felting or matting of crystals.

The coating in the burning zone protects the refractory lining from wear and improves the thermal insulation of the kiln wall. As a rule, therefore, coating is a desirable feature. However, under adverse conditions it may become too thick or, as already mentioned, form objectionable rings. There are various methods of removing such excess features.

– removal by manual breaking-out with the aid of rods and pneumatic hammers;
– by melting or spalling off the accretions with the flame;
– by quenching them with low-pressure or high-pressure water jetting, causing them to break up;
– dislodging by shooting them with projectiles from a special gun ("industrial cannon");
– bursting them with the aid of carbon dioxide cartridges, more particularly the Cardox method.

The conditions of application and other information on these methods are given in Tables 4 and 5. The cost figures in the latter table relate to 1970. An added advantage of the pump supplying a high-pressure water jet is that it can be used also for other purposes in the cement works, such as awkward cleaning jobs, etc.

Table 4: Suitability of methods for the removal of objectionable coating

nature of coating	methods and means employed: 1. melting or spalling off with the flame	2. low-pressure water jetting	3. high-pressure water jetting	4. shooting with special gun	5. bursting by Cardox method	6. other methods
coatings in preheater	–	–	(+)	–	–	by hand with pneumatic hammer
meal rings	(+)	–	–	(+)	(+)	festoon chains as internal fittings
slurry rings	–	–	–	–	–	
sinter rings	+	+	+	+	+	
clinker rings	+	+	+	+	+	by hand with rods
coatings in inlet of cooler	–	–	+	–	+	by hand with rods or with special equipment
movable material agglomerations	–	(+)	(+)	+	–	

Table 5: Cost and effort demanded by methods for the removal of a sintering (based on 1970 price levels)

effort/cost	methods and means employed: 1. melting or spalling off with the flame	2. low-pressure water jetting	3. high-pressure water jetting	4. shooting with special gun	5. bursting by Cardox method	6. removal by hand with pneumatic hammer and rods
downtime	reduced output for 10–20 hours	3 to 6 times 0.5 hour with intermediate heating	3 to 6 times 0.5 hour with intermediate heating	0.5 to 1 hour	0.5 to 1 hour	2 to 4 days
manpower required	none	6 to 8	1 or 2	2 or 3	2 or 3	more than 3 men for about 1 day
initial cost, overheads	none	low	DM 15000 to 18000	about DM 7000	DM 8000, incl. filling machine about DM 15000	none
cost per ring	according to loss of production	according to loss of production increased fuel consumption		500 DM to 2000	100 DM	same as 2. and 3.

With 4. and 5. the loss of production is small because of short downtime

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IV. Clinker storage

By B. Kohlhaas

1 General

It is still fairly common practice to store cement clinker in outdoor stockpiles or in roofed, but not completely enclosed, buildings. The dust nuisance associated with such open storage is accepted as something to be put up with. However, with increasing size of present-day cement works and the widespread introduction of environmental pollution prevention regulations the need to accommodate the clinker in properly enclosed storage structures is growing more urgent.
The recommended storage capacity to be provided corresponds to between 15 and 30 days' production.

2 Forms of construction and space requirements

Various forms of construction have been introduced, depending on the planned storage capacity, the subsoil and the general local conditions. In every case, however, it is essential to have the greatest possible effective capacity and to ensure efficient emptying. The possibility of storing different types of clinker may have to be considered.
Some commonly encountered types of storage structure, with their theoretical effective capacities and space requirements, are shown in Figs. 1 a, 1 b and 1 c.

Comments on Fig. 1 a:

In the design of large silos for clinker it is always important to find the most efficient and inexpensive shape. The various clinker silos indicated in Fig. 1 a are all of approximately the same capacity, the object of these drawings being to show various possibilities (though without laying any claim to completeness). The dotted lines represent the discharge arrangements.
Silo type "a" occupies little space on plan, but is very unfavourable because it has a large lateral surface area and, on account of the high ground bearing pressure, requires an expensive foundation slab. Besides, the bucket elevators are very high.

Type "b" represents a widely applied form of silo construction, which compares favourably with the preceding type in having a smaller lateral surface area, lower bucket elevators, and simpler foundation, especially on structurally favourable soils.
Silo type "c" has its discharge outlet above ground level, enabling the clinker to be loaded directly into vehicles. Against this, however, the cost of construction is higher than for type "b".
Thanks to its large diameter (50 m), silo type "d" is characterized by its small surface area in relation to its cubic capacity. The cost is nevertheless considerable.
The forms of construction represented by types "e" and "f" appear favourable because there is no expensive foundation slab and the bucket elevators are of limited height. However, silo "e" is not to be recommended in the event of high ground-water level and difficult soil conditions, especially if excavation presents problems. Silo "f" is more favourable because of its shallower foundation, but it requires two tunnels or ducts for withdrawing the clinker.
The segregation of the clinker into larger and smaller particles, which is liable to occur in silos of large diameter, can be compensated by means of a receiving tunnel, provided with a clearing plough, under the silo.

Comments on Figs. 1 b and 1 c:

Comparisons between silos in respect of their capacity should be based on the effective capacity, namely, the quantity of clinker that can be discharged freely, i.e., without requiring the assistance of a bulldozer-type vehicle. In this sense silos have

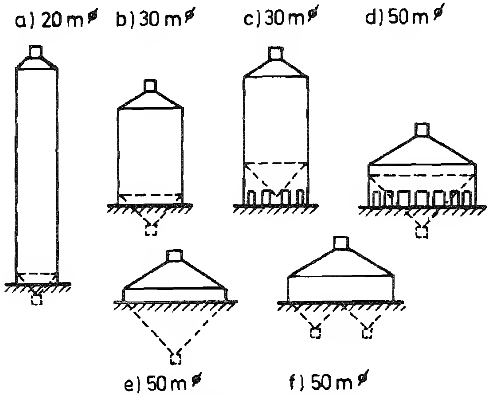

Fig. 1 a: Various forms of silo for the storage of cement clinker (capacity about 30 000 m³) (from Funke, 1968)

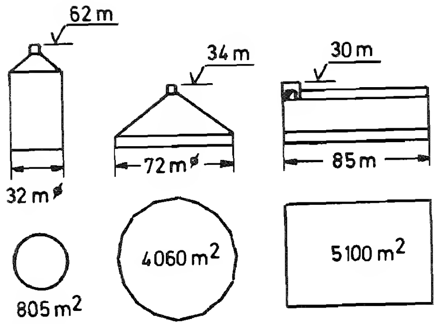

Fig. 1 b: Comparison of space occupied by silo, tent-shaped circular store and hall-type building for 45 000 t of cement clinker (from Sillem, 1972)

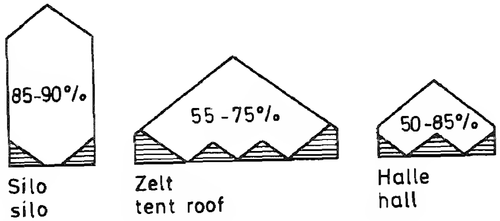

Fig. 1 c: Comparison of effective capacities of clinker storage structures (from Sillem, 1972)

the largest effective capacity, as appears from the comparative diagrams in Fig. 1 c representing a silo, a tent-roof (conical) building and a hall-type building. In all three cases a flat bottom has been assumed. In the "tent" and the "hall" the clinker lying in the dead space can, however, be pushed to the discharge openings by bulldozers or other means.
Round silos occupy the least space, as Fig. 1 b shows. Against this is the drawback that large silos exert high ground bearing pressures and are therefore unsuitable on structurally poor soil. In such cases there remains a choice between the "hall" and the "tent". The latter may then be the less expensive alternative on account of its smaller area on plan in a case where a piled foundation is required. Another possibility on sites with difficult soil conditions is to build a number of small silos instead of one large silo.
Some clinker storage structures actually built are shown in the following illustrations.

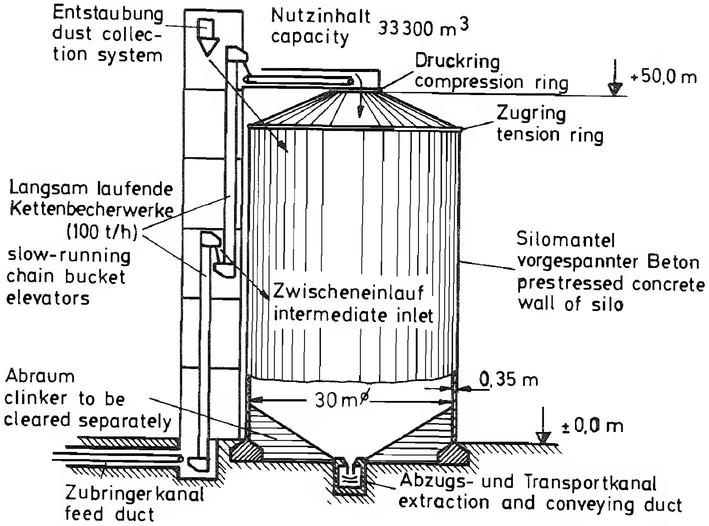

Fig. 2: Silo constructed of prestressed concrete for 50000 t of clinker (from Funke, 1968)

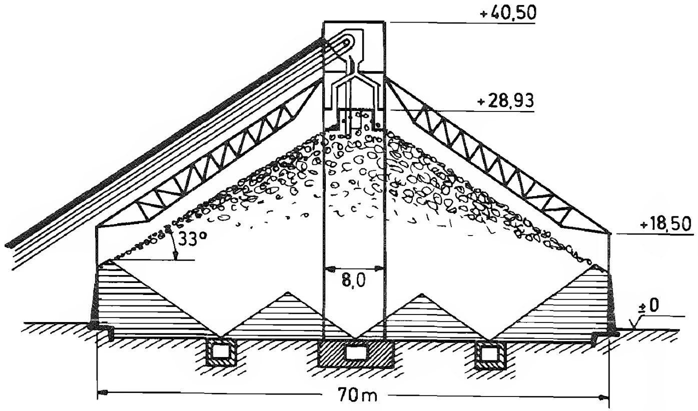

Fig. 3: Cross-section through a tent-shaped clinker store (from Sillem, 1972)

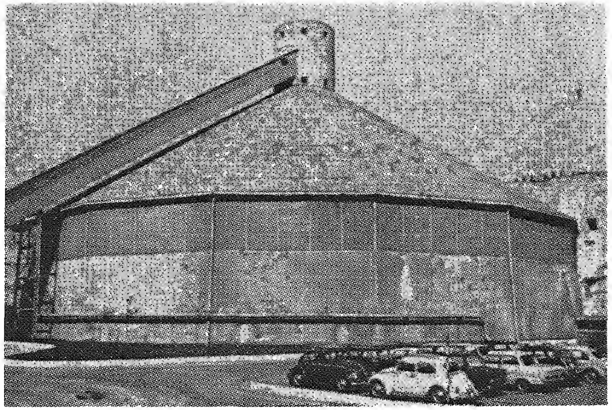

Fig. 4: View of the storage building shown in cross-section in Fig. 3 (from Sillem, 1972)

A prestressed concrete silo of about 50 000 t capacity is illustrated in Fig. 2. Clinker is fed into it by two low-speed bucket elevators and is discharged through four bottom outlets.

Figs. 3 and 4 show a "tent" clinker storage building of the Enci cement works at Maastricht, Holland. It has a capacity of 70 000 t. The lower part is of regular 16-angle shape on plan, with a diameter of 70 m. The tent-shaped roof is supported at its centre on an internal silo capable of holding 2000 t of clinker. An inclined conveyor feeds clinker into the silo and storage building. Extraction is effected through 26 bottom openings with three belt conveyors.

3 Selection criteria

The following considerations are applicable in deciding what type of storage structure to use and what capacity it should have

- What fluctuations in demand for the product, depending on the state of the market, will the storage facilities have to cope with?
- Should a larger quantity be stored as an operational safeguard, more particularly to cope with fluctuations in production?
- What is the structural quality of the subsoil on which the structure is to be built?
- Are there statutory regulations for environmental protection to be complied with?
- Is the storage structure to be built of steel or of reinforced concrete?
- What methods of filling and emptying it are envisaged?

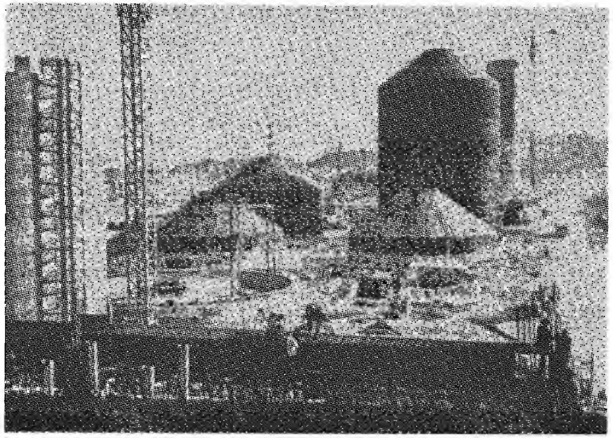

Fig. 5: Steel silos for clinker storage; erection by the spiral method (from Peter/Erni, 1978)

For example, if the subsoil is unfavourable (low bearing capacity, etc.), but steel plate is obtainable at relatively low cost, it will be possible to build inexpensive clinker storage silos by means of the rational "spiral" method of erection. The construction material should be weather-resistant steel (grade WT52-III conforming to EW 087-70, acceptance under test certificate 50049-3.1 C) (Fig. 5).
Steel silos of appropriate construction require no painting and are therefore virtually maintenance-free.
In every project involving the selection of a particular type of silo or other storage structure it is highly advisable to make a careful cost comparison.

4 Design

Besides the temperature of the clinker there are also some other important aspects to consider in connection with the design of steel or reinforced concrete silos. It would be outside the scope of this book to go into these aspects of structural design. Further guidance is obtainable from, among others, the following publications:

H. J. Klischat, Lengerich/Westf., ZKG 25/1972/494–495.
I. Kleine, Heidelberg, ZKG 25/1972/391–394.
K. Pieper, P. Martens, D. Kroll and K. Wagner, Techn. Univ Brunswick, ZKG 23/1970/337–342 (whether further literature references are given).
K. Hering, Brunswick, ZKG 28/1975/523–525 (with further references).

5 Filling and emptying silos and other storage structures

With regard to the filling and emptying of silos for cement clinker it is necessary to ensure that these operations are accomplished as smoothly and regularly as possible so as to avoid unequal (one-sided) loads on the wall and foundation.
The familiar material handling appliances – such as vertical bucket elevators, inclined bucket elevators, belt conveyors, deep-pan conveyors and steel apron conveyors – are used for delivering clinker to silos and storage buildings and for uniformly distribution it in these structures. As the clinker coming straight from the cooler may still have a fairly high temperature (up to about 350° C), great care must be taken to choose handling devices of the right type and adequate capacity for the purpose.
Discharge of clinker from storage structures is effected through the appropriate number of outlets equipped with rotary valves or gates (as closing devices) with attached vibratory chutes feeding the clinker to apron conveyors, belt conveyors or clinker handling cars.

6 Storage buildings and outdoor stockpiles

6.1 Storage buildings

Besides silos, which have already been more particularly described in the preceding chapters, various types of building classifiable as "halls" or "sheds" are used for the storage of cement clinker. Older structures of this category, originally of semi-open construction, can usually be subsequently closed in. However, the cranes with which such buildings are often equipped and which serve to distribute and reclaim the clinker will then generally have to be replaced by handling devices which raise much less dust. In addition, efficient dust collecting equipment will have to be installed.
New large clinker storage buildings are designed as fully enclosed structures and are equipped with appropriate handling installations. Operation is automatic, requiring no attendant personnel.
One of the largest enclosed hall-type clinker storage buildings is at Rørdal cement works, Aalborg, Denmark (Fig. 6). Its dimensions on plan are 240 m × 60 m and its capacity is 200 000 t. The roof covering consists of aluminium panels.
Other examples of clinker storage buildings are illustrated in Figs. 7, 8 and 9.
The storage installation shown in Figs. 10, 11 and 12 is a special case. Soil conditions and the ground-water level on this site made it possible to build a structure with considerable effective capacity (about 100 000 t), approximately 120 m long, 55 m wide and 34 m high. The whole stockpile is roofed by a lightweight covering consisting of galvanized steel profiled sheeting supported on precast reinforced concrete beams. The clinker is fed to the store by means of a deep-pan conveyor. Short apron conveyors withdraw the clinker from seven outlet openings discharging into a handling duct.

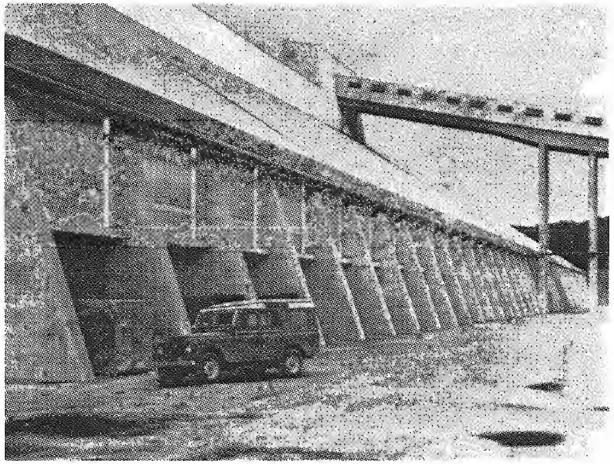

Fig. 6: **Clinker store** (from Christensen, 1971)

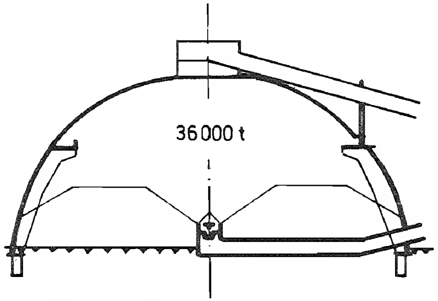

Fig. 7: Clinker storage building at Port-la-Nouvelle cement works, France (from Smith/Homassel/Juan, 1978)

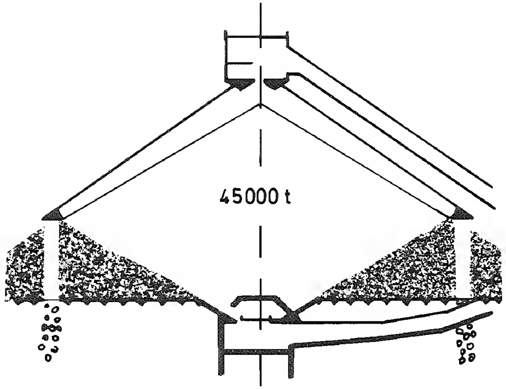

Fig. 8: Clinker storage building of a cement works at Boulogne-sur-Mer, France (from Smith/Homassel/Juan, 1978)

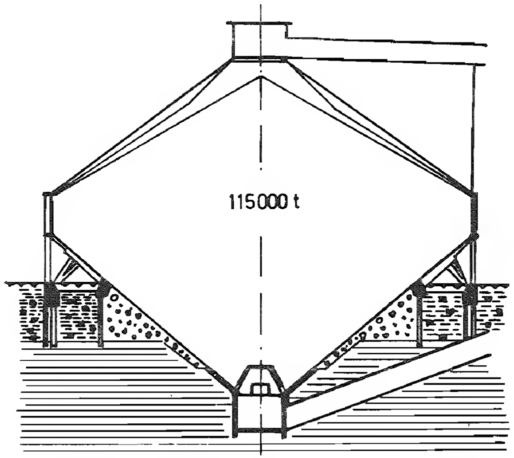

Fig. 9: Clinker storage building of a cement works at Montreal, Canada (from Smith/Homassel/Juan, 1978)

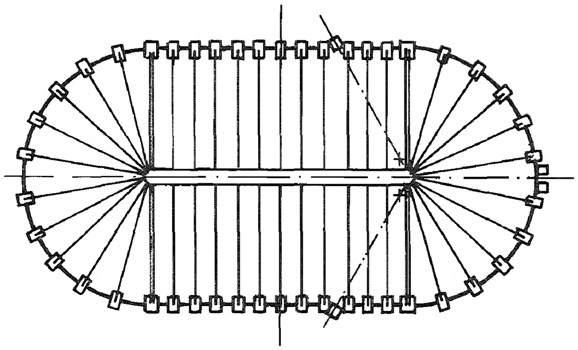

Fig. 10: Plan of clinker store (from Kühle, 1974)

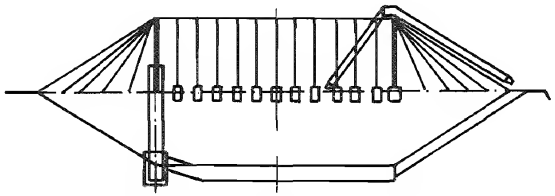

Fig. 11: Longitudinal section through clinker store (from Kühle, 1974)

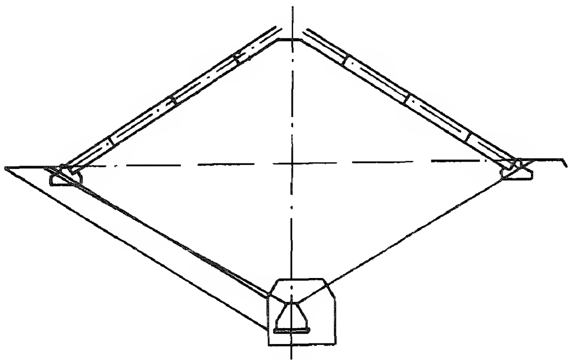

Fig. 12: Cross-section through clinker store (from Kühle, 1974)

6.2 Outdoor stockpiles

Where environmental conditions permit it, outdoor stockpiling of clinker is an alternative form of storage. In that case, however, precautions to prevent excessive dust formation are necessary. Some possible ways of overcoming dust nuisance more particularly in connection with depositing the clinker on the pile are indicated here.

On no account should the clinker be dropped from any appreciable height under open (non-enclosed) free fall conditions onto the stockpile. Reclaiming from the pile should be by underfloor extraction through a suitably large number of discharge openings. With such arrangements the dust-raising methods of clinker handling, e. g., by using bulldozers to push it to the openings, can largely be obviated. Steel or reinforced concrete columns or towers for depositing the clinker on the stockpile may be constructed. These structures are provided, for example, with discharge openings at various levels, closed by gates or flaps, which can be opened as required to let out the clinker falling from the top of the tower (Figs. 13 and 14).

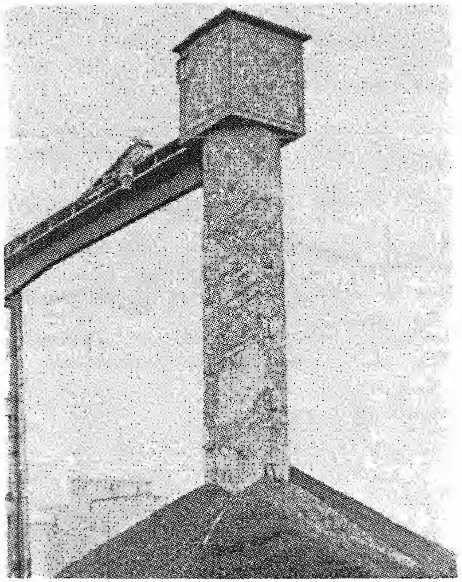

Fig. 13: Concrete tower on outdoor clinker stockpile (from Funke, 1968)

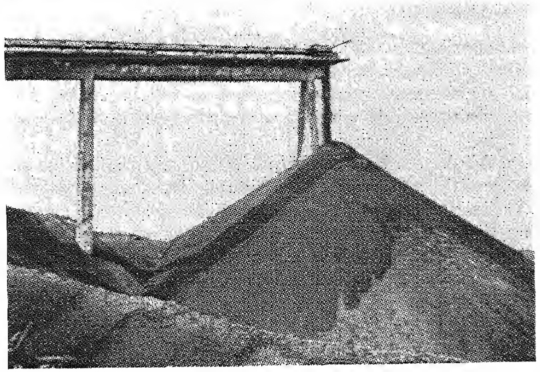

Fig. 14: Steel tubular structure on outdoor clinker stockpile, at end of belt conveyor (from Funke, 1968)

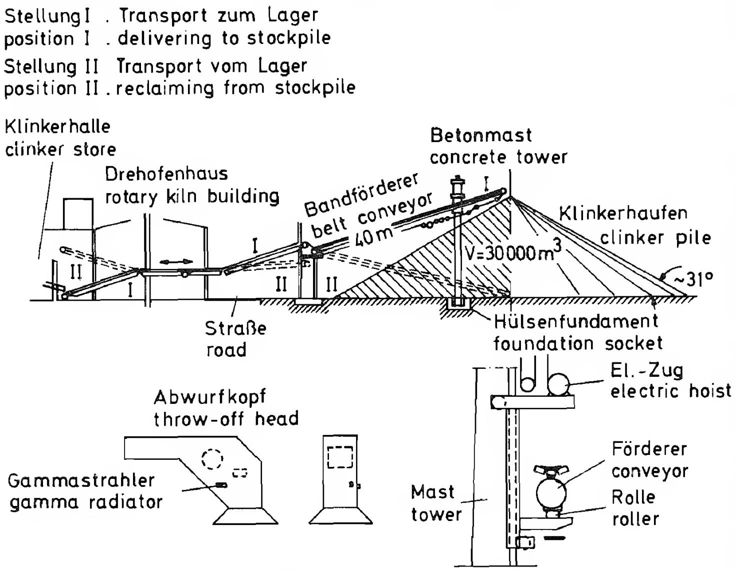

Fig. 15: Outdoor clinker stockpile with belt conveyor that can be raised and lowered (from Funke, 1968)

Alternatively, stacker belt conveyors mounted on booms which can be raised and lowered may be used for stockpiling the clinker. In such installations the boom movements are automatically controlled by means of sensors responding to the height of the pile. A storage system of this kind is shown in Fig. 15.

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V. Cement silos

By H. K. Klein-Albenhausen

1 General

Installations for the storage of cement are an important feature of a modern cement works. With the steady increase in size of the production units there has been a corresponding increase in silo capacity. Large-capacity silos with diameters of 20 m and more, and capable of holding anything up to 30000 t of cement per silo, must be so designed that they can be efficiently emptied. Also, expensive intermediate handling of the cement should be eliminated as far as possible in order to have suitably rationalized procedures for despatching the cement from the works.

The number and size of the cement silos will depend on the operating and despatch conditions of the works in question In the past, silos were preferably installed at ground level, but in recent years the trend towards elevated construction has been manifest, the advantage being that it is then often possible to feed the cement direct – i.e., without intermediate handling – from the silo to the sack filling machines or to the bulk despatch loading bays.

The present article will not be concerned with the various devices for the emptying of bins, hoppers, etc. but will deal only with the bottom discharge arrangements used in modern big cement silos.

2 Large-capacity silos

The silos used for the storage of cement present no problems as regards filling them. To achieve equally problem-free emptying is much more difficult. Modern cement silos are invariably equipped with pneumatic handling systems and pneumatic discharge and flow regulating devices. Integral features of such silos comprise open trough-type conveyors and/or aerating units supplied with compressed air for enlivening the cement to assist its discharge from the silo. The air is supplied by rotary piston blowers at pressures ranging from about 4000 to 8000 mm w.g. Depending on the silo emptying system, the air supply rate for attaining a certain cement discharge rate varies. In some systems the compressed air admitted into the silo is partly discharged through venting pipes, while in others all the air is discharged along with the cement.

A principle that all pneumatic emptying systems have in common is that of partial aeration of the silo floor or bottom. The object is to introduce only so much compressed air into the silo as is needed for discharging the cement at the desired rate and to keep the energy consumption as low as possible.

The air blown into the silo is automatically switched cyclically from sector to sector of the bottom by a special distributing device in conjunction with pneumatically operated valves which are opened and closed as required.

The silo bottom should be so designed as to be free from any slopes or other features that are liable to cause "bridging". Vertical walls and short handling paths at the bottom of the silo are two basic requirements for efficient emptying. With fairly long paths it may occur that "dead" zones are formed in which the cement remains stagnant and may eventually harden, necessitating subsequent removal by manual labour. Important, too, is that all the aerating sectors of the silo should be activated uniformly in succession, in order to ensure that the material level goes

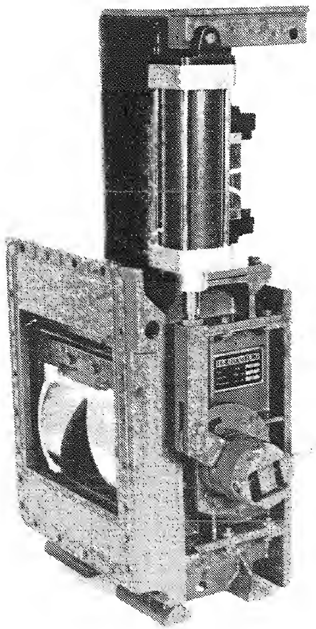


Fig. 1: Flow control gate, pneumatically operated
(IBAU Hamburg)

Fig. 2: Lump crusher with dual drive

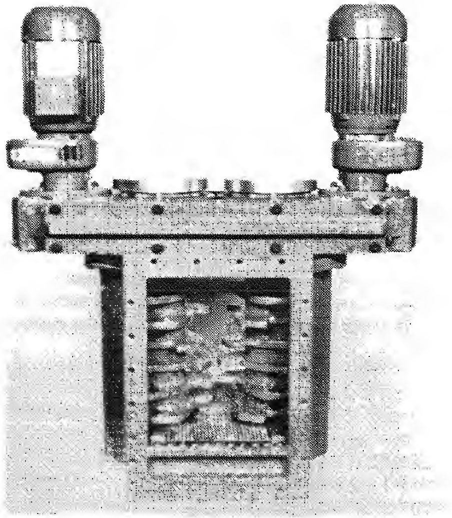
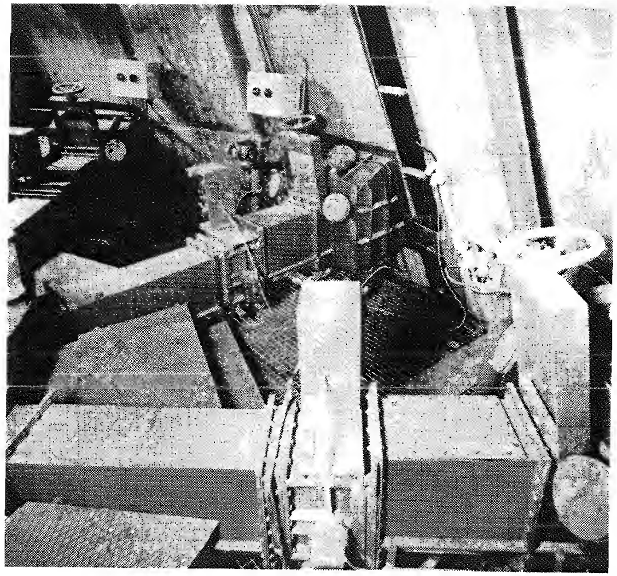


Fig. 3: Use of compressed air lances to assist discharge



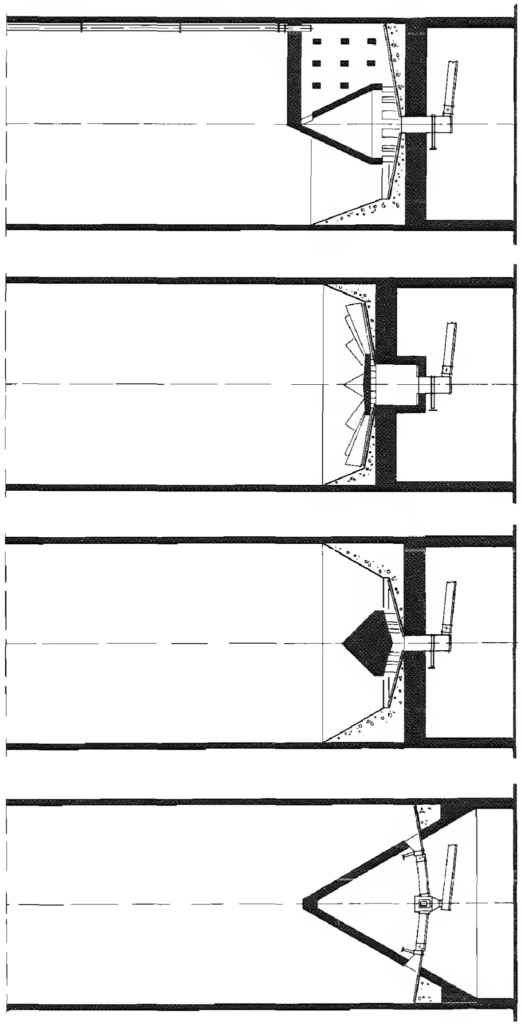


Fig. 4: Various types of silo bottom construction

down at a regular rate over the entire silo cross-section. Failure to achieve this may give rise to funnelling, so that some of the contents will rush to the outlet, while in other parts there is little or no motion of the cement, which is thus liable to solidify and harden there.
Flow regulating valves control the rate of cement discharge from the silo (Fig. 1). To prevent blockage of the outlet in the case of cements tending to form lumps, disintegrators for breaking the lumps may be installed ahead of these valves (Fig. 2). Large lumps which cannot be dealt with in this way have to be broken up from outside the silo with the aid of compressed air lances (Fig. 3).
Fig. 4 shows various forms of construction used for silo bottoms in the present-day cement industry.

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E. Packing and loading for despatch

I. Packing

1 Introduction

What type of packaging is chosen for the despatch of cement will depend on local circumstances and on the most favourable transport facilities available: road, rail or water (ship or barge).
In principle, there are three possibilities: packing in sacks*) (or bags), despatch in bulk, and "big bag" despatch.

*) The terms "sack" and "bag" are synonymous in the present context; "bag" is the preferred term in American publications.

2 Types of packaging

2.1 Sacks

Cement used to be packed in wooden casks or steel drums. Nowadays jute or paper sacks are used, especially the latter. The paper valve sack is the type most extensively employed. In contrast with the ordinary open mouth sack the valve sack is closed on all sides except for a small opening at one corner through which the cement is introduced into the sack. As a result of the excess pressure that develops inside the sack, this opening automatically closes (in the manner of a non-return valve) on completion of the filling operation.
These sacks are mostly of the so-called pasted end type, but sacks with sewn end closures are still used to some extent.
Paper sacks for cement are generally of two-ply construction, consisting of kraft paper made from soda pulp, each ply having a weight of 90 – 100 g/m^2. For rough handling conditions, sacks with three, four or five plies may be used.

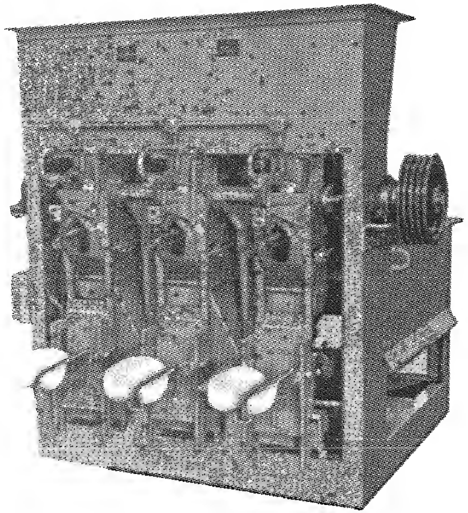


Fig. 1: Three-spout packer. Operated by one man, this machine can fill about 900 pasted-end paper sacks per hour, each sack containing 50 kg of cement

Special machines are required for filling valve sacks. These machines are commonly called sack (or bag) packing machines or merely "packers".
Sewn valve sacks are more awkward to fit onto the filling spouts of the machine, besides requiring more attendant personnel to achieve equal filling rates, than pasted valve sacks.

2.1.1 In-line packing machines

Packers of the in-line type, with three or four stationary filling spouts side by side, are now used only for capacities of up to 80 t/hour. The type more extensively used, with capacities ranging up to 120 t/hour, is the rotary packer (Figs. 1 and 2).
The cement fed to the packer is passed through a screen which stops any foreign bodies, oversize particles or lumps (above 3 mm in size) which are undesirable in the cement and a possible hazard to the machine.
The level of the cement in the feed hopper over the machine must be kept as nearly constant as possible.

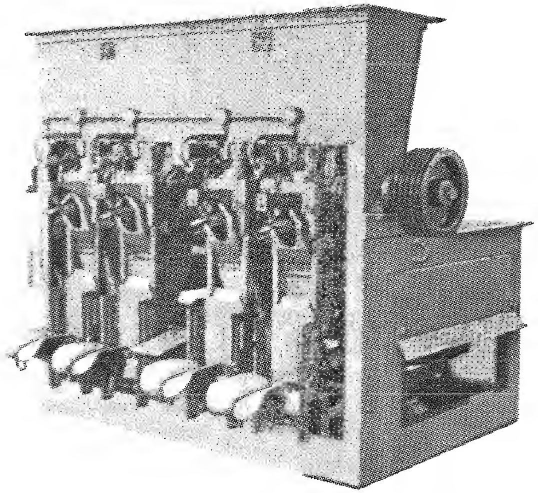


Fig. 2: Four-spout packer for about 1100—1200 sacks per hour

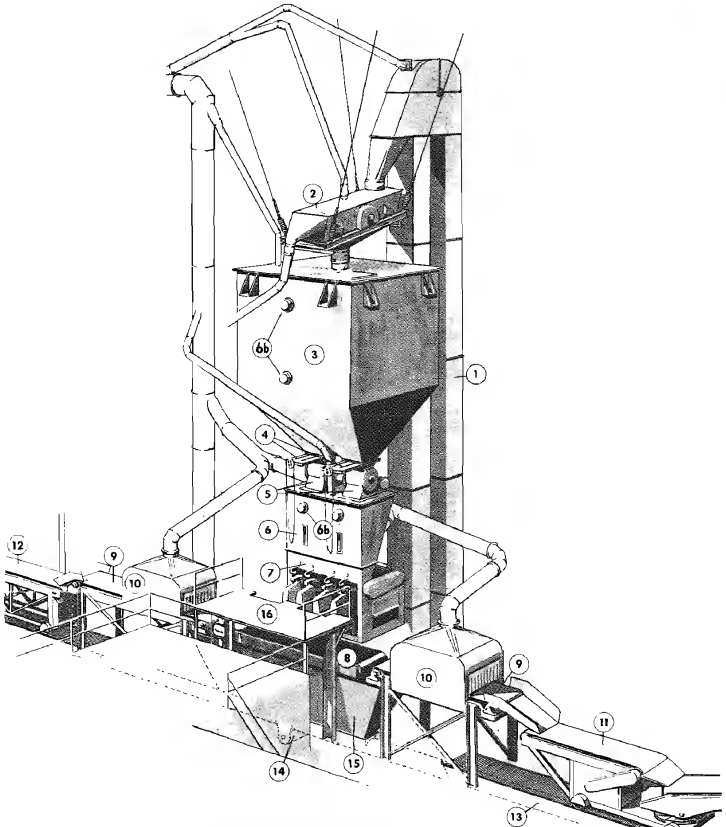


Fig. 3: Main features of a packing plant

1 bucket elevator
2 vibratory screen
3 storage hopper
4 gate valve
5 rotary valve
6 packing hopper
7 packing machine
8 flat belt conveyor
9 flat belt conveyor
10 sack cleaning unit
11 loading belt for road vehicles
12 loading belt for railway wagons
13 spillage return screw
14 spillage return screw
15 spillage collecting hopper

The most favourable arrangement for a sack packing installation is as follows (Fig. 3):

- bucket elevator;
- screen to retain oversize (vibrating screen);
- storage hopper with minimum and maximum level indicators;
- rotary valves controlled by the material level over the packing machine;
- packing hopper;
- packing machine;
- spillage hopper for returning cement, spilled in the sack filling operation, to the packing circuit.

The feed bucket elevator must on no account be followed by pneumatic handling devices which fluidize the cement, as this would adversely affect the packing operation.
To eliminate dust nuisance, dust extraction intakes connected to collecting equipment of ample capacity should be provided at all points where dust is especially likely to arise. The "dust" (i.e., cement) collected in this equipment is returned to the packing circuit.

1 equal-arm weigh-beam
2 weight box
3 upper guide rod
4 lower guide rod
5 sack support
6 filling spout
7 damping system
8 fine feed control device
9 proximity switch
10 proximity switch

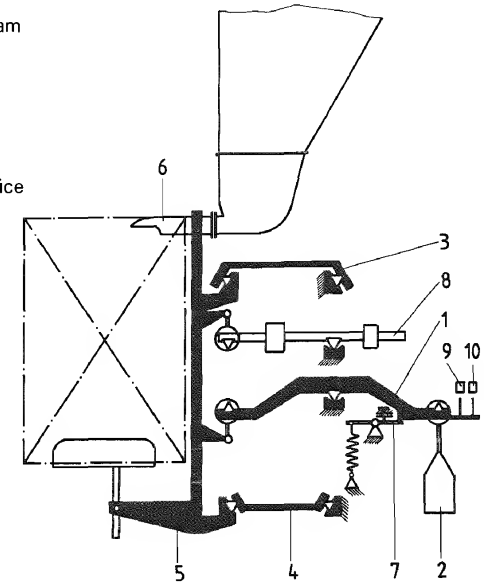


Fig. 4: Weighing system of an in-line packing machine with equal-arm weigh-beam

The filled sacks are conveyed, by the shortest possible routes, to the despatch loading bays. The number of transfer points from one one belt conveyor to another should also be as few as possible, for at each transfer the sack receives a jolt, causing its contents to shift over to one side. As a result the sacks become somewhat lopsided in thickness and more difficult to stack. Arrangements to achieve smooth jolt-free transfer from one belt to the next will avoid abrupt changes in level, e.g., by means of curved transition belts.
Figs. 4 and 5 illustrate the weighing system of the in-line packen.

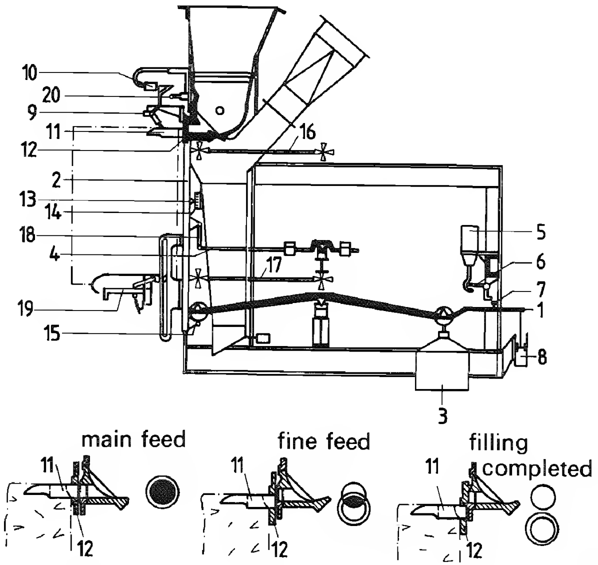


Fig. 5: Diagram of an in-line packing machine with equal-arm-weigh-beam

1	weigh-beam	11	filling tube
2	sack support	12	filling tube plate
3	weight box	13	pointer
4	trickle feed regulator	14	scale
5	magnet	15	suspension for weigh-beam
6	catch	16	upper guide rods
7	engaging hook	17	lower guide rods
8	damper	18	suspension for fine feed control
9	sack holder	19	saddle
10	switch for magnet	20	roller

2.1.2 Rotary packers

In contrast with the in-line packer with its filling spouts mounted stationary side by side, requiring the machine operator to move from spout to spout, the spouts on the rotary packer move one by one into position in front of the operator, who merely has to fit the valve sacks onto them as they successively pass him. Figs. 7 and 7a show the weighing systems conventionally used for rotary packers, while Fig. 8 schematically shows an electronic system. There are rotary packers with 6, 8, 12 and 14 spouts.
Besides higher capacity, the rotary packer has the advantage that the period between depositing the successively filled sacks on the belt conveyor is constant, so that they are equally spaced on the conveyor, a fact which is especially advantageous when sacks have to be stacked by hand.
Manual application of valve sacks to the spouts.
It has hitherto been common practice to fit the sacks by hand onto the filling spouts. With in-line machines an operator can attain rates of 300 sacks per hour in this way. With an eight-spout rotary packer he can attain approximately the same rate. Much will depend on his skill and experience, however. Some performance figures are given in Table 1.

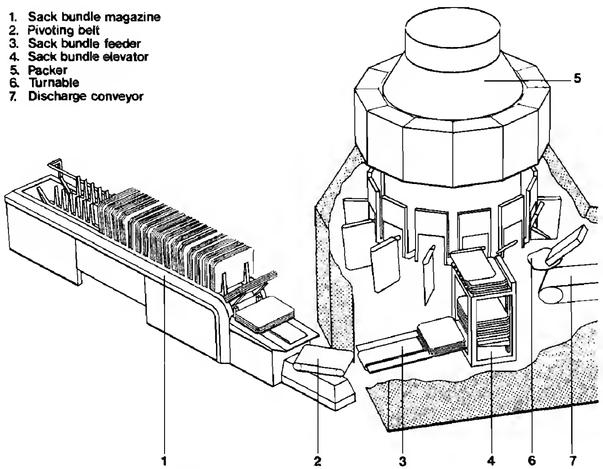


Fig. 6: Automatic Rotating Packer

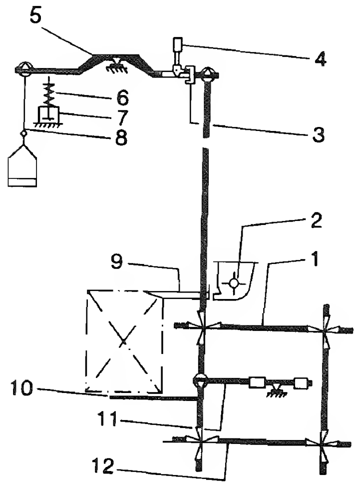


Fig. 7a: Diagram of a rotary packing machine with equal-arm weigh-beam

Description:
Eight such weighing units are mounted together on a turnable and are jointly served by a storage hopper. Each unit comprises an equal-arm weigh-beam (5), the weight box (8), the support for carrying the valve sacks (sack saddle) which is guided by the guide rods (1 and 12), the fine feed control device (11), the damping system (6 and 7), and the quick-action cut-off system (3 and 4).

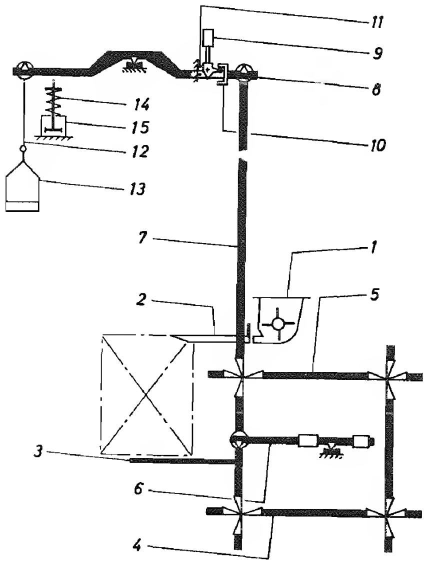


Fig. 7b: Diagram of a rotary packing machine with equal-arm weigh-beam

Description (Fig. 7b): see p. 485

1 filling spout
2 valve sack
3 sack saddle
4 sack support
5 compression spring (overload protection)
6 mounting frame
7 flexural bar
8 flexural bar attachment
9 machine frame
10 weight indication
11 guide rod
12 tare relieving spring
13 tension adjusting screw
14 fixing aperture
15 stop
16 stop screw

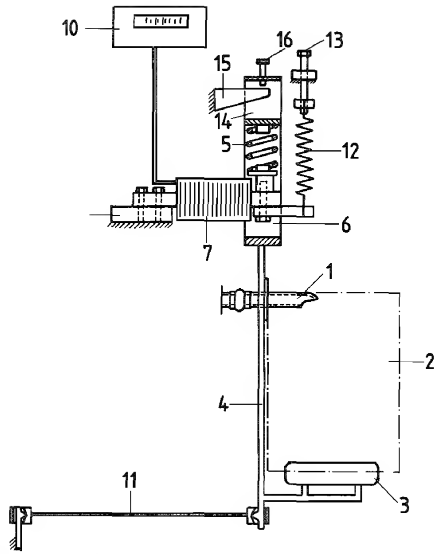


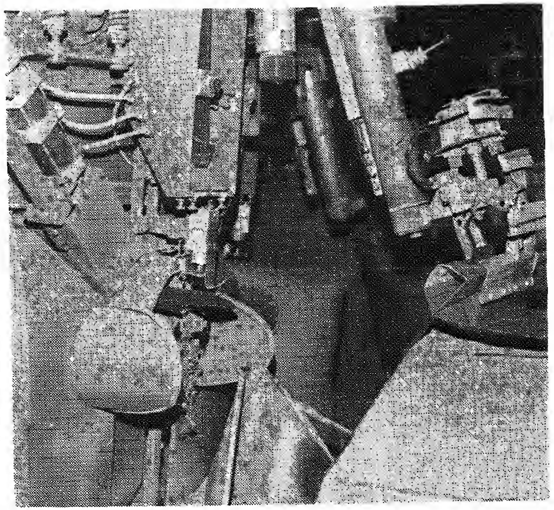
Fig. 8: Diagram of an electronic weighing system

Description (Fig. 7b):
The cement is discharged by the impeller (1) through the filling spout or tube (2) into a valve sack standing on the saddle (3), which is guided parallel by the rods (4 and 5). The load is transmitted from the saddle through the tension rod (7) to the equal-arm weigh-beam (8) comprising two levers interconnected by a cross-piece (10). The weight box (13) is suspended from the weigh-beam. The damper (15) functions only when the left arm of the beam descends. For quickly cutting off the feed flow when the specified filling weight has been attained, the upper flange of the channel-section cross-piece (10) causes the small weight (9) to tilt, so that its attached lever now suddenly swings down and strikes the lower flange of the cross-piece and thus accelerates the closure of the feed device. The spring (14) on the damper assists this action. On removal of the load, the right arm of the weigh-beam rises, and the lower flange of the cross-piece swings the small weight (9) back to its upright position of rest against the stop (11).
Up to eight of these weighing units can be assembled on a single turnable for a rotary packer. Maximum load per unit: 50 kg.

Table 1: Rotary packer data: filling time, diameter and number of filling spouts (Haver & Boecker, Oelde/Westf.)

filling time	time per revolution	speed	circumferential velocity		time/sack 8 spouts	time/sack 6 spouts	packing rate 8 spouts	packing rate 6 spouts
			Ø 2000	Ø 1600				
(sec)	(sec)	(r.p.m.)	(m/sec)	(m/sec)	(sec)	(sec)	(sacks/h)	(sacks/h)
7	9.8	6.17	0.65	0.518	1.22	1.63	2940	2200
8	11.2	5.4	0.568	0.456	1.4	1.86	2570	1930
9	12.6	4.8	0.505	0.403	1.57	2.08	2290	1730
10	14.0	4.3	0.45	0.362	1.75	2.34	2060	1540
11	15.4	3.91	0.41	0.328	1.93	2.58	1870	1400
12	16.8	3.6	0.378	0.303	2.1	2.81	1710	1280
13	18.2	3.32	0.338	0.278	2.32	3.1	1550	1160
14	19.6	3.08	0.324	0.259	2.45	2.28	1470	1100

an angle of 260° for filling has been assumed



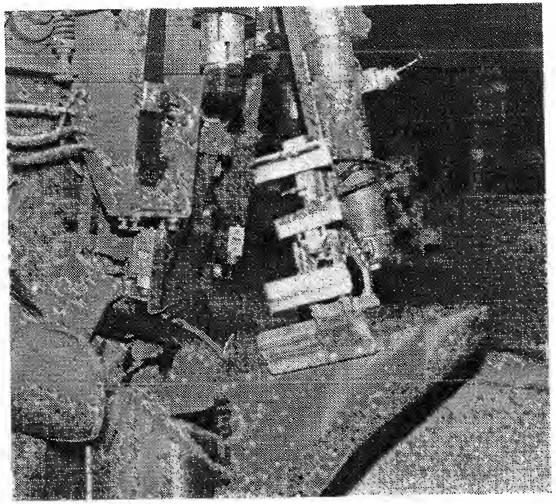


Fig. 9 and 10: Automatic sack applicator

2.1.3 Fully automatic operation

The fully automatic sack applicator functions independently of the human operator, whose physical effort it relieves, leaving him merely to perform a supervisory function (Figs. 9 and 10).
The automatic applicator comprises a stationary part and the applicator arms on the rotary packing machine. Each sack is taken individually from a reel of sacks and is lifted by means of suction cups, which also open the valve of the sack. Each spout

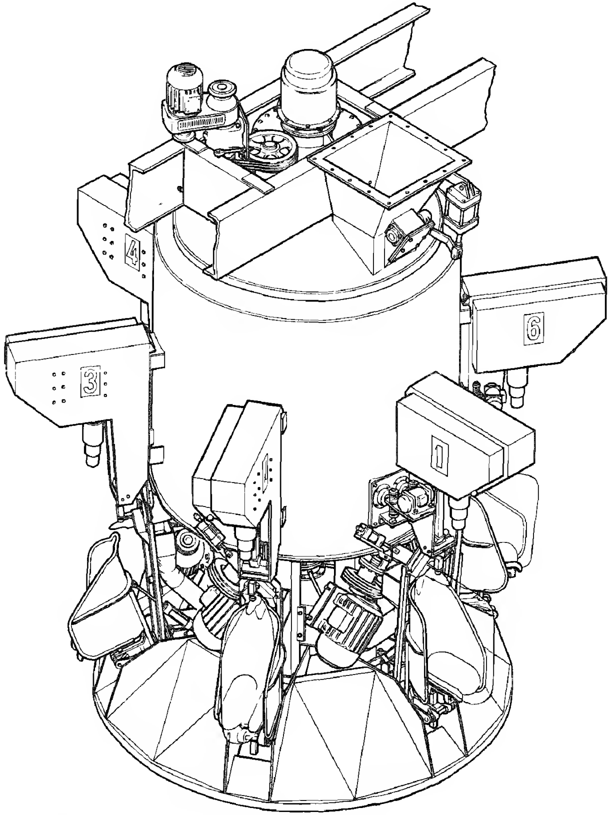


Fig. 11: Rotary packer of compact design

of the packer is provided with a swivel arm (rotating with the machine) fitted with a gripping device which seizes a sack and swings it into position over the spout as soon as the previous sack has been filled and released from the spout.
The automatic system does, however, suffer from a disadvantage: whereas the human operator can smooth out any crumpled sacks before applying them to the spouts, the automatic applicator is unable to do this. Hence it is essential to ensure that the empty paper sacks are supplied uncrumpled to the packer. Fig. 11 illustrates a rotary packer of compact design.

2.1.4 Sack magazine

Hitherto standard practice has been to combine the empty valve sacks into bundles of 25, stacked in an interlocking configuration to form 1000-sack bales held together by steel straps. This method of packaging the empty sacks presents no problems so long as they are applied manually to the filling spouts of the packing machines. With the advent of the automatic applicator a different form of packaging for the empty sacks had to be devised, however.
The following procedure has been found satisfactory: In the sack factory the empty sacks are laid overlapping one upon another and wound on a spool. The reel of

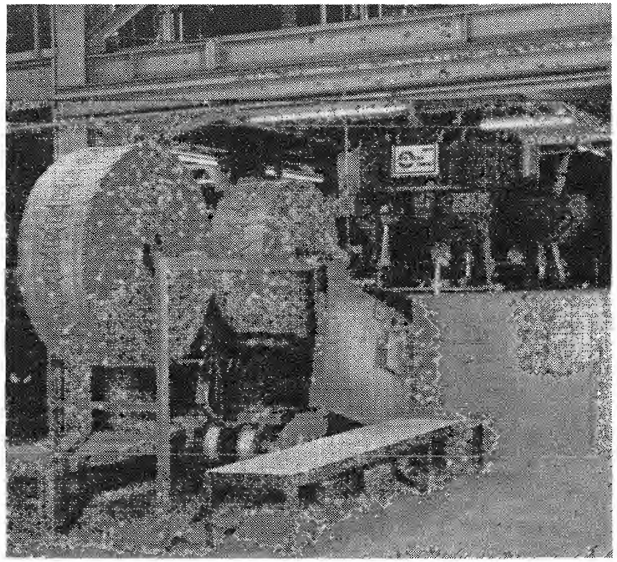


Fig. 12: Paper sack reels

sacks formed in this way is held together by means of two strips of plastic which are likewise wound into the reel. Up to about 3000 two-ply sacks can thus be assembled into a reel, usually of 1500 mm diameter, which is sufficient to keep the average packing machine supplied with sacks for about $1^1/_2$ hours.
The reels of sacks are delivered to the cement works, where they are each mounted on a spindle and placed on the unreeling stand (Fig. 12).
Experience has shown the reel to be the best form of sack magazine, because with this method the sacks do not crumple; indeed, they are smoothed in the reeling-up operation. Besides, convenient and easy separation of the sacks is possible only with the overlapping arrangement.

II. Despatch of cement

The ratio of the quantities of cement despatched in sacks to those despatched in bulk varies greatly from one country to another and from one part of the world to another. For the individual cement works the respective proportions of "sack" and "bulk" are dictated by outside circumstances and cannot be notably altered. There is, however, scope of choice with regard to the type of loading equipment to be used and the degree of automation of the loading and despatch operations.
So-called "big bag" despatch is as yet of minor importance in comparison with despatch in sacks and in bulk.

1 Despatch in sacks

Products packed in sacks are despatched either as loads consisting of individual sacks and formed with the aid of loading machines or as palletized unit loads, i.e., each consisting of a number of sacks stacked and secured on a pallet. Palletizing of the sacks can take place either directly on the floor of the despatch vehicle itself or indirectly for intermediate storage. The loading of sacks from intermediate storage may vary in the method of supporting and of securing the unit loads.

1.1 Individual sack loading

Machines for the loading of sacks individually into road vehicles or railway waggons are used in circumstances where fully automated loading systems are not appropriate to requirements or offer no advantages. The wide variety of sack loading machines that have been developed over the years can be subdivided in principle into those for loading open vehicles and covered vehicles respectively. Also, a distinction can be made between machines for side loading, for rear-end loading and for loading "from above", more particularly from the upper storey of the sack packing house (Fig. 13).
For loading sacks into railcars, machines comprising three main sections and mobile in three dimensions are employed, as illustrated in Fig. 14.

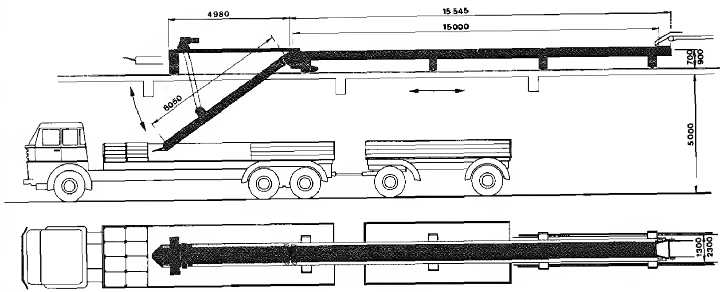


Fig. 13: Loading machine for open vehicles

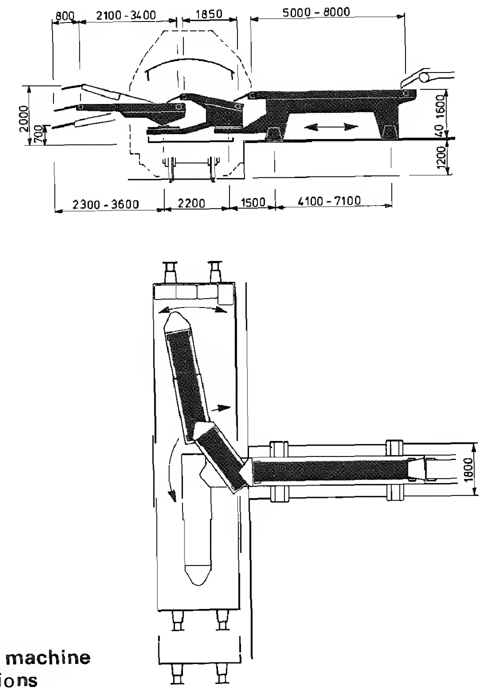
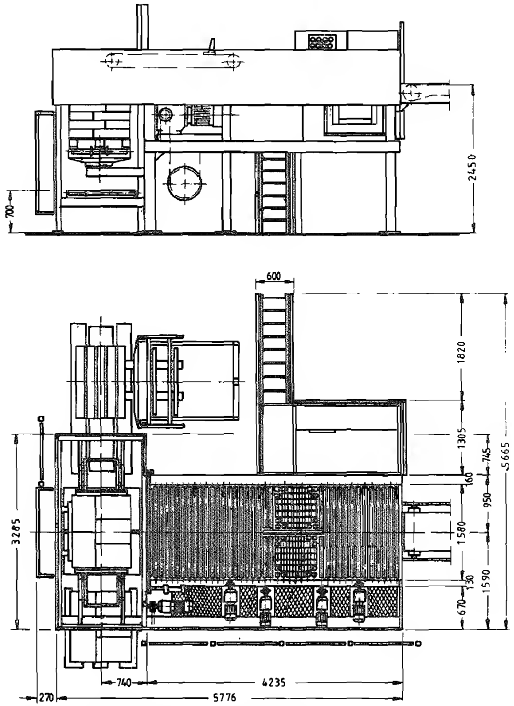


Fig. 14: Railcar loading machine mobile in three dimensions

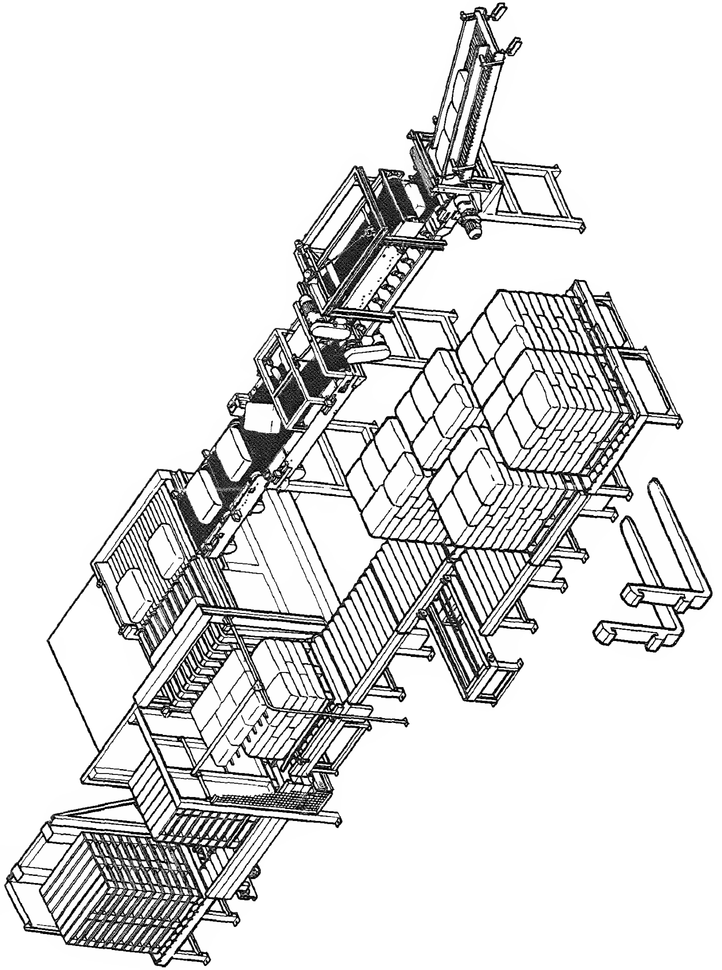
Despite the degree of mechanization, sack despatch with the aid of such loading machines is labour-intensive. As an approximate guide, a gang of at least two men can attain a loading rate of about 1500 sacks per hour.

1.2 Palletizing

In cases where conventional systems of individual sack loading are ruled out, palletizing is often applied, i.e., the sacks are assembled into unit loads (stacks) on pallets by machines. These loads are placed in intermediate storage, from where they are removed with the aid of fork-lift trucks and loaded into vehicles for despatch. In the storage areas, the palletized loads have to be stacked



Figs. 15a and b: High-capacity automatic palletizers for the building materials industry, embodying different design principles



one upon another, and this means that these loads themselves have to consist of stable stacks of sacks held securely in place. The standard sack used in the cement industry has dimensions of 600 mm × 400 mm × 130 mm and is usually stacked five to a layer (with dimensions of 1000 mm × 1200 mm) on so-called pool pallets (800 mm × 1200 mm) or on ISO pallets (1000 × 1200 mm). The size of the palletized unit load is determined by the number of layers of sacks. Four layers form a load weighing 1 ton. Loads comprising from four to eight sacks are commonly employed. They are handled usually by fork-lift trucks equipped with double-length forks, so that pallets can be handled two at a time. For this more efficient operating procedure, it is best to employ trucks with 7.5 t lifting capacity. As a rule, automatic palletizers are required to attain rates of 2000 to 2400 sacks per hour in order to enable these machines to operate directly in-line with modern high-capacity sack filling machines which operate at similar rates. Examples of automatic palletizers of such capacity are shown in Figs. 15a and b.

In conjunction with the further development of high-capacity rotary sack packing machines, automatic palletizers have been developed to a high level of technical performance, enabling palletizing rates of up to 5000 sacks per hour to be attained. Substantial savings in terms of capital expenditure on buildings and handling appliances can be effected by the use of such machines.

For making comparisons between direct palletizing on vehicles and the use of stationary automatic palletizers producing palletized loads for intermediate storage, it will be useful to summarize the advantages offered by these two alternative systems

With direct palletizing, the sacks are transferred directly from the packer to the vehicle, on the floor of which the empty pallets, provided by the customer, are placed in readiness to receive the sacks of cement. The obvious advantage of this system is that the space and cost of construction required by a storage building are saved. Also, less personnel is needed than when palletized loads have to be put into, and reclaimed from, intermediate storage, and the expense of handling empty pallets and repairing damaged ones is likewise eliminated. Against this, the stationary palletizer producing palletized loads for intermediate storage has the advantage that the stored loads form a buffer stock which makes the cement works and/or the customer less closely dependent on the available sack packing and palletizing capacity. It also enables the packers to be operated on a single-shift basis and yet to meet peak demands from customers by using more fork-lift trucks to load their vehicles when circumstances require this. Besides, with loading palletized sacks from store, there is a high degree of flexibility in assembling a mixed load – e.g., different types or grades of cement – on one and the same vehicle. A rule of thumb for estimating the required intermediate storage capacity is that it should be able to contain between two and four times the daily quantity despatched.

1.3 Direct loading

Direct loading means the placing and stacking of sacks directly on the floor of the vehicle by means of automatically functioning machines.

Modern machines of this kind operate on the same principle as automatic palletizer, i.e., they stack the sacks in a regular interlocking pattern and thus form a carefully assembled load with adequate stability. From the technical point of view, these automatic loading machines embody different modes of operation. In one type of machine, the individual sacks, or a whole layer of sacks, are lifted with the aid of suction cups and lowered by the action of hydraulically powered telescopic arms onto the floor of the vehicle. Another type of machine has electromechanical operation: an automatic machine of this kind, for direct loading onto open vehicles, is shown schematically in Fig. 16.

A fully automatic machine for the rear-end loading of sacks into covered vehicles or into containers has been developed. It consists essentially of a telescopic jib and a palletizing (stacking) head, the whole installation being mounted on a transverse travel unit, so that it can serve several vehicle loading bays located side by side and parallel to one another (Fig. 17).

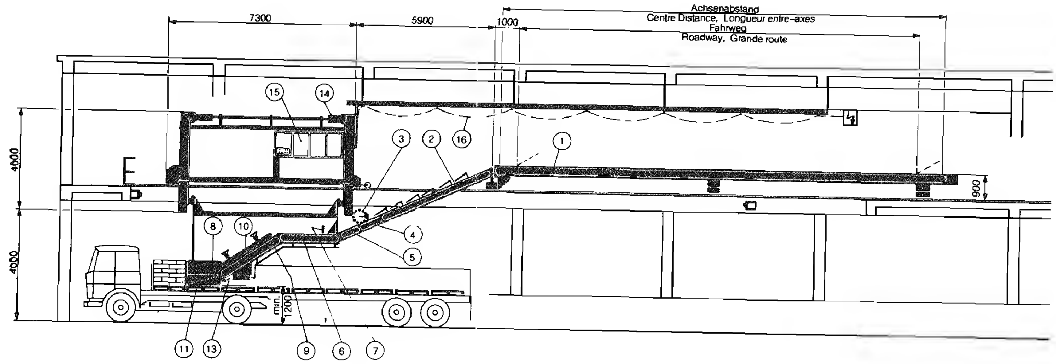
Because of the prevailing climatic conditions, a high proportion of covered vehicles is used for cement despatch in Western and Northern Europe. Such vehicles, and also open ones with fixed superstructural features (e.g., on-board cranes), can most suitably be loaded with side loading machines travelling at ground level. These deposit the sacks in layers equal in width to the width of the vehicle by means of a retractable fork extending sideways over the floor of the vehicle. After each layer has been placed, the loading fork is raised a distance equal to one layer depth, and the next layer is then formed on the previous one. When the predetermined number of layers has been loaded, the machine travels a certain distance (parallel to the longitudinal direction of the vehicle) equal to the stacking width it can serve from each working position. It then lowers its fork and starts loading the first layer of the next stack on the vehicle floor, and so on.

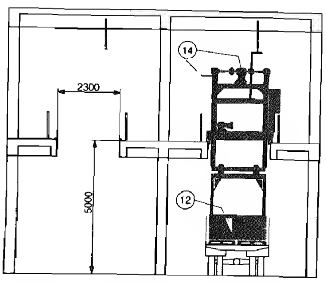
Automatic loaders for sacks are at present built for nominal loading rates of up to 2500 sacks per hour Idle time due to vehicle changing or to switching from one sack packing machine to another can be reduced by the interposition of buffer sections, i.e., sections along the handling path where filled sacks are temporarily accumulated in order to smooth the irregularities in supply from the packers or in demand from the loaders.

For the loading of sacks into railcars, only partially automated systems have as yet become available.

2 Bulk loading

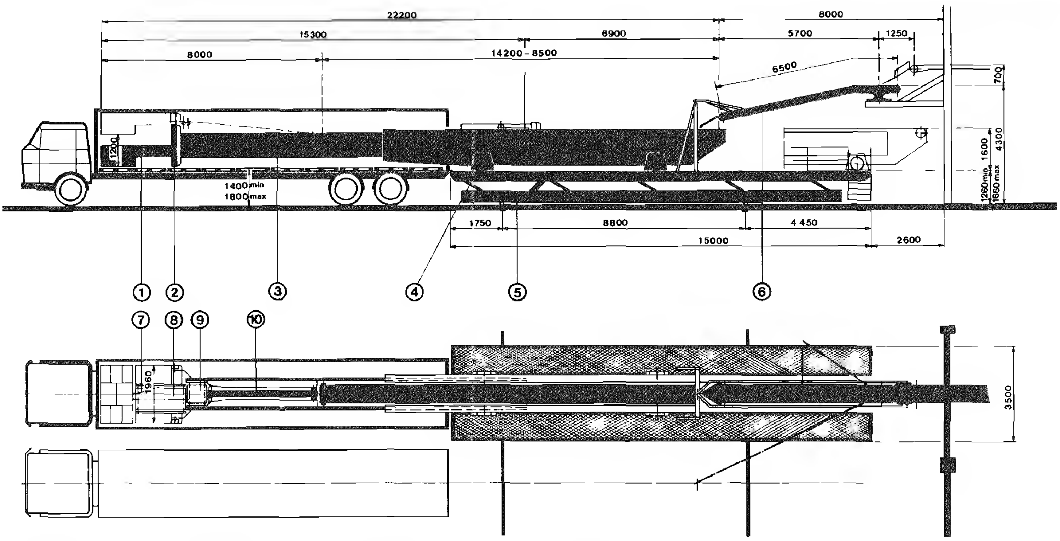
Despatch of materials in bulk offers better possibilities for automation of the material flow than does the despatch of unit loads. This is reflected in the design features and arrangements for bulk loading in the cement industry. An important requirement applicable to such bulk loading installations is that they must enable the cement to be fed into tanker-type bulk carrier vehicles under dust-free conditions. Handling rates for the bulk loading of road and rail vehicles should range up to about 400 t/hour, while ship or barge loading installations usually have





① verfahrbarer Flachgurtforderer	① Travelling Flat Belt Conveyor	① Transporteur mobil
② Staustrecke	② Storage Section	② Transporteur incliné
③ Sackbugelwalze	③ Press Roller	③ Cylindre à repasser les sacs
④ Stauband	④ Storage Belt	④ Bande inclinée
⑤ Abzugsband	⑤ Take Off Belt	⑤ Bande extractrice
⑥ Richtband	⑥ Positioning Belt	⑥ Bande d'alignement
⑦ Sackdrehvorrichtung	⑦ Bag Turner	⑦ Dispositif de pivotement des sacs
⑧ Stapelkopf	⑧ Loading Head	⑧ Tête de chargement
⑨ Zuteilband	⑨ Feed Belt	⑨ Tapis de régulation
⑩ Sackbugelband	⑩ Press Belt	⑩ Bande à repasser les sacs
⑪ Rollentisch	⑪ Roller Table	⑪ Table à rouleaux
⑫ Abschieber – positionieren	⑫ Slide Positioner	⑫ Positionneur à translation
⑬ Andrückbalken	⑬ Pressplate	⑬ Poussoir
⑭ Hubwerk	⑭ Winch	⑭ Moteur de levage
⑮ Schaltschrank mit Steuertafel	⑮ Control Panel with Instrument Board	⑮ Armoire électrique avec tableau
⑯ Stromeinspeisung	⑯ Power Cables	⑯ Câble d'alimentation

Fig. 16: Diagram illustrating the principle of an automatic sack loader, with electromechanical action, for open vehicles



① Palettierkopf	① Palletizing head	① Tête de palettisation	① Cabeza paletizador
② Hubwerk	② Lift mechanism	② Système de levage	② Elevador
③ Teleskopausleger	③ Telescopic beam	③ Eléments télescopiques	③ Pluma telescópica
④ Fahrwerk	④ Chassis	④ Système mobile - avancer-reculer	④ Mecanismo de traslación
⑤ Wagen für Querfahrt	⑤ Pull-off conveyor	⑤ Système mobile pour déplacement latérale	⑤ Movimientos transbordador
⑥ Zubringerband	⑥ Feed-in conveyor	⑥ Bande d'amenée	⑥ Cinta alimentadora
⑦ Sackdrehvorrichtung	⑦ Bag turner	⑦ Pivoteur de sacs	⑦ Dispositivo de giro del saco
⑧ Positionierband	⑧ Positioning belt	⑧ Bande de positionnement	⑧ Cinta posicionadora
⑨ Abzugsband	⑨ Accumulating conveyor	⑨ Bande d'espacement	⑨ Cinta dosificadora
⑩ Staustrecke	⑩ Storage section	⑩ Transporteur de retenue	⑩ Cinta acumuladora

Fig. 17: Automatic rear-end loader for containers or covered vehicles

to attain substantially higher rates (1000–1200 t/hour). The equipment must be simple to operate, so that it can be worked by the vehicle drivers themselves.
A well regulated and continuous supply of the bulk material is obviously essential to the proper functioning of a bulk loading system. The handling and feeding devices for conveying the cement to the actual loading equipment comprise rotary gates, screw conveyors, flow regulating valves, vibratory troughs, chain conveyors, belt conveyors, etc.

2.1 Loading installations

The principal feature of a bulk loading system is the loading unit (Fig 18) comprising the inlet casing with dust extraction ports, the double-bellows loading spout (alternatively, a telescopic steel tube may be employed), the conically tapered nozzle which fits into the inlet opening of the bulk carrier vehicle to form a dust-tight seal, and the filling level monitor. The double-bellows spout made of textile fabric can suitably be used for the handling of non-abrasive bulk materials. For abrasive materials, or where high throughput rates are required, the telescopic steel tube is more appropriate. The dust-laden air displaced from the interior of the vehicle's bulk carrying tank during the filling operation escapes through the annular space between the inner and the outer tube of the double-bellows spout (or between the telescopic steel tube and the bellows-type outer tube in which it is enclosed) and is extracted by suction. In this way any dust pollution of the environment is obviated.
The level monitoring devices used for bulk loading systems are – depending on the nature of the material handled – based on one of various operating principles: mechanical, capacitive or inductive level sensing.
Bulk loading installations as described here are either of the stationary or the mobile type. In the latter, which may in turn be of the swivelling or the travelling variety, the loading unit is connected to a movable feed system. The vehicle to be loaded is moved into position in the loading bay and need then not be moved again until the filling operation has been completed. The loading spout is successively moved to the several filling inlets along the vehicle. The movable feed system supplying the spout is of various types, depending on the nature of the bulk material being handled: screw conveyors, airslides (of single or articulated construction), telescopic tubes, etc. Mobile bulk loading installations are shown in Figs. 19a and b.
All the loading units may either be connected to a central dust filter or each be equipped with an individual dust filter forming an integral feature of the loading unit (Fig. 20). If different grades or types of cement are loaded by means of the same installation, the individual-filter system avoids mixing of the different dusts collected from the installation. The dust can therefore be returned to the material flow, with the further advantage that pipes or ducts from the loading units to a central filter are dispensed with. The air extraction rate associated with the filling of bulk carrier vehicles ranges from 1000 to 3000 m³/hour.
As the actual loading operation starts and proceeds fully automatically as soon as the nozzle of the loading spout has been fitted to the inlet of the vehicle, the scope

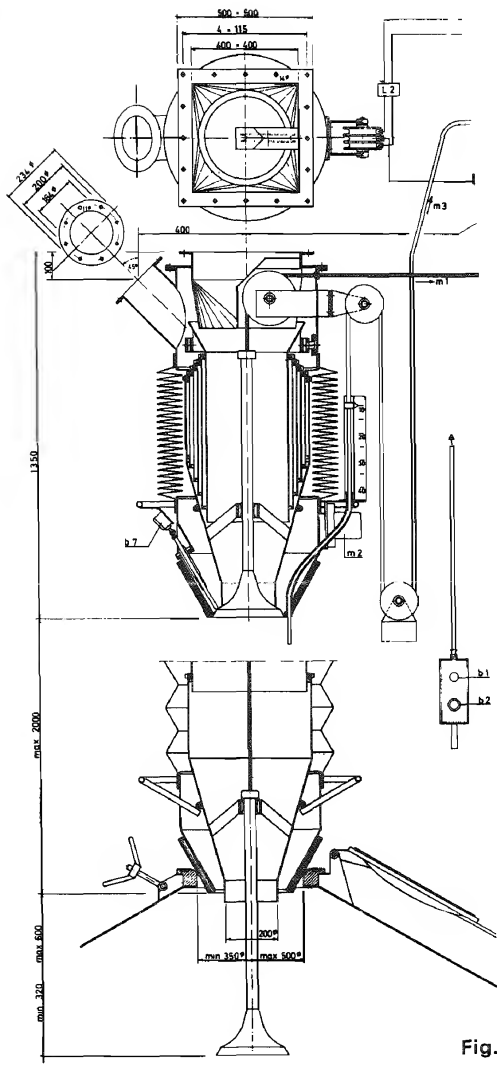
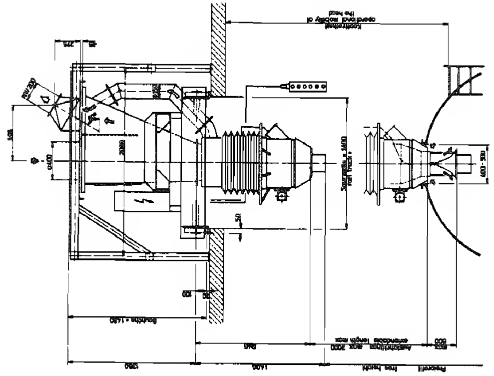
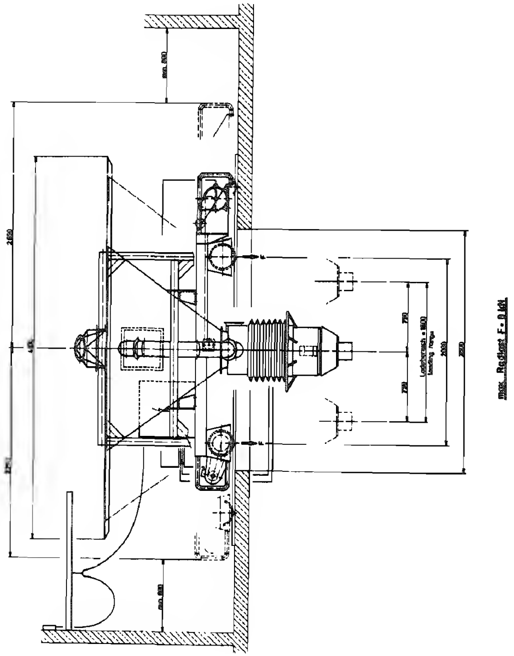
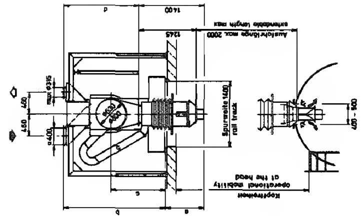
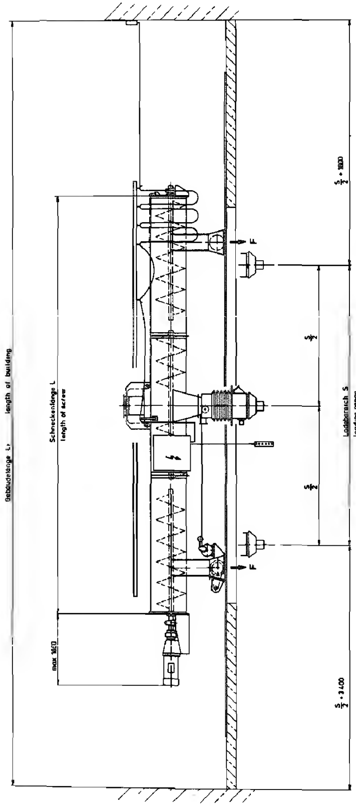


Fig. 18: Bulk loading unit









	Ladebereich S loading range	Gebäudelänge L₁ length of building	Schneckenlänge L length of screw	Radlast F=kN
Schnecke Ø630, max. Förderleistung 200 m³/h; screw max conveying capacity; a 1300, b 1925, c 900, d 1825	3000	11200	4500	12
	4000	13200	5500	13
	5000	15200	6500	15
	6000	17200	7500	16
	7000	19200	8500	18
	8000	21200	9500	20
	9000	23200	10500	22
	10000	25200	11500	24
	11000	27200	12500	26
	12000	29200	13500	28
	15000	35200	16500	33
Schnecke Ø800, max. Förderleistung 300 m³/h; screw max conveying capacity; a 1300, b 2175, c 1000, d 2075	3000	11200	4500	16
	4000	13200	5500	18
	5000	15200	6500	20
	6000	17200	7500	23
	7000	19200	8500	25
	8000	21200	9500	28
	9000	23200	10500	30
	10000	25200	11500	33
	11000	27200	12500	35
	12000	29200	13500	38
	15000	35200	16500	46

Figs. 19a and b: Mobile bulk loading installations with ranges of up to 1.5 m (a) and 12 m (b)

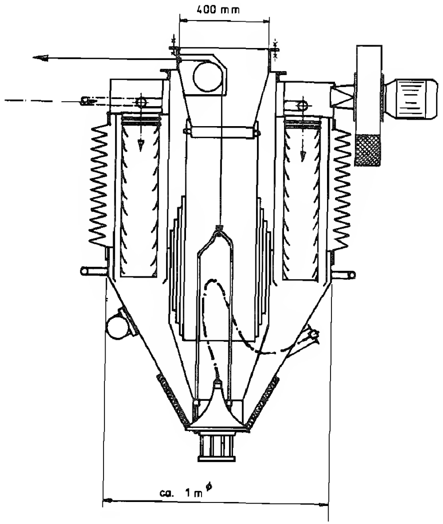


Fig. 20: Loading unit with integrated dust filter

for further rationalization of bulk loading operations is somewhat limited. There are still possibilities in further developing the "self-service" operation of the loading equipment by the vehicle drivers, together with the issuing of vehicle identification badges to achieve time-saving automation of the despatch documentation and all other records required in connection with the commercial transaction.

2.2 Weighing systems

Various weighing systems are used in conjunction with the bulk despatch of materials.
With net weighing the loading spout is preceded by a calibratable weigh hopper in which a predetermined quantity of cement is held in readiness for discharge into the vehicle. The tare weight of the vehicle is therefore irrelevant. Alternatively, a predetermined quantity of cement can be discharged, as required, from a weigh hopper which is constantly kept filled. A cut-off switch on the loading spout prevents overfilling.
With gross weighing the vehicle stands on a calibratable weighbridge during the loading operation. First the tare weight of the empty vehicle is determined. Filling

then commences, and when a certain predetermined fill quantity has been reached, the material supply is cut off and the gross weight of the vehicle plus its load is measured. The net weight is then obtained as the difference between gross and tare. A cut-off switch in the loading spout prevents overfilling.
In an alternative method the tare weight of the empty vehicle is determined on a calibratable weighbridge, and the vehicle is then filled from a non-calibratable weigh hopper preceding the loading spout. Finally, the gross weight of the loaded vehicle is measured on a calibratable weighbridge.

3 Loading of clinker and crushed stone

The equipment for the bulk loading of cement, as already described, is generally suitable for dealing with pulverized materials possessing good flow properties. For the despatch in bulk of coarsely granular or lump materials such as clinker, crushed stone, lump lime and other comparable products, loading installations as shown in Fig. 21 are used. The installation in question comprises a telescopic tube, a dust arresting dome, bellows-type tubes for dust removal, and the suspension system for the loading unit. During the loading operation it must be ensured that the outer rubber apron is kept constantly in contact with the conical pile of material, to prevent escape of dust. A level monitoring device in the dome emits signals which cause the loading unit to be progressively raised as the pile grows higher.
The loading unit for tanker-type bulk carrier vehicles can be fitted with a dust arresting dome to make it suitable for the loading of open vehicles (Fig. 22) It thus becomes a dual-purpose system suitable for either type of vehicle. The dome is connected by means of quick-action clip-on devices to the lower end of the unit. By means of a special catch the cut-off valve in the spout is held open during the loading of open vehicles. This form of construction of the loading unit is often also used for ship or barge loading.

4 "Big bag" despatch

This method of cement despatch from the plant has been made possible by the development of extremely strong and durable packaging material and the use of appliances that can handle loads weighing 1 ton and more. Thanks to these arrangements, very large sacks (called "big bags") can be used for transporting the cement to its destination.
More particularly, two systems are to be distinguished in connection with the "big bag" despatch of cement one-trip (disposable) sacks and re-usable sacks.
The disposable packaging consists of a large square-bottom sack made of a plastic ribbon fabric, lined with plastic sheet. During the filling operation the mouth of the sack is held wide open. On completion of this operation the sack is closed by

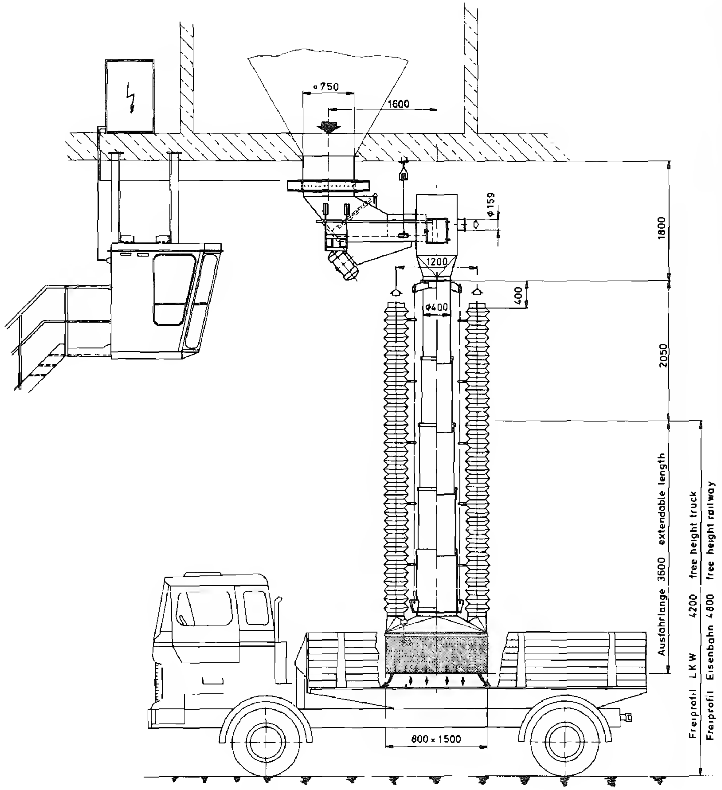


Fig. 21: Loading of clinker or crushed stone

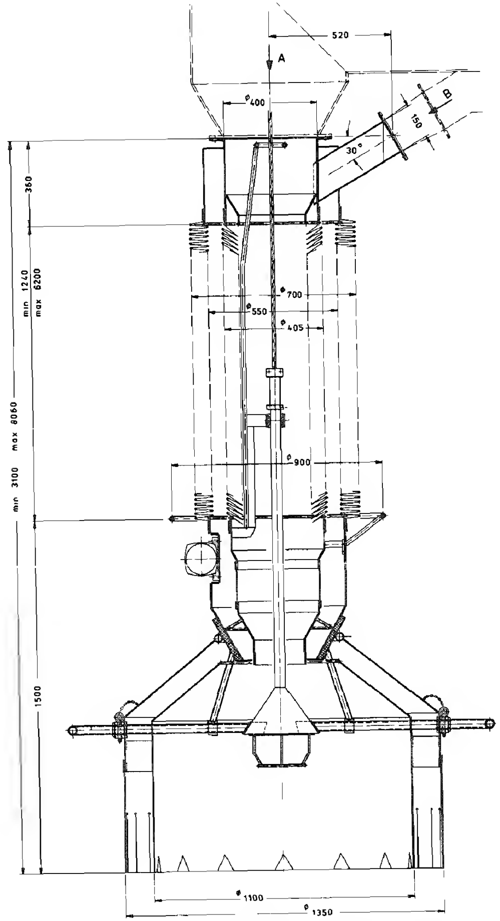


Fig. 22: Dual-purpose loading unit for tanker-type and for open vehicles

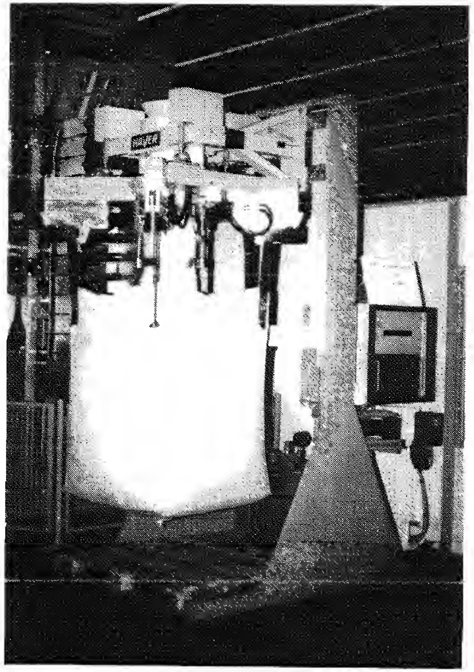


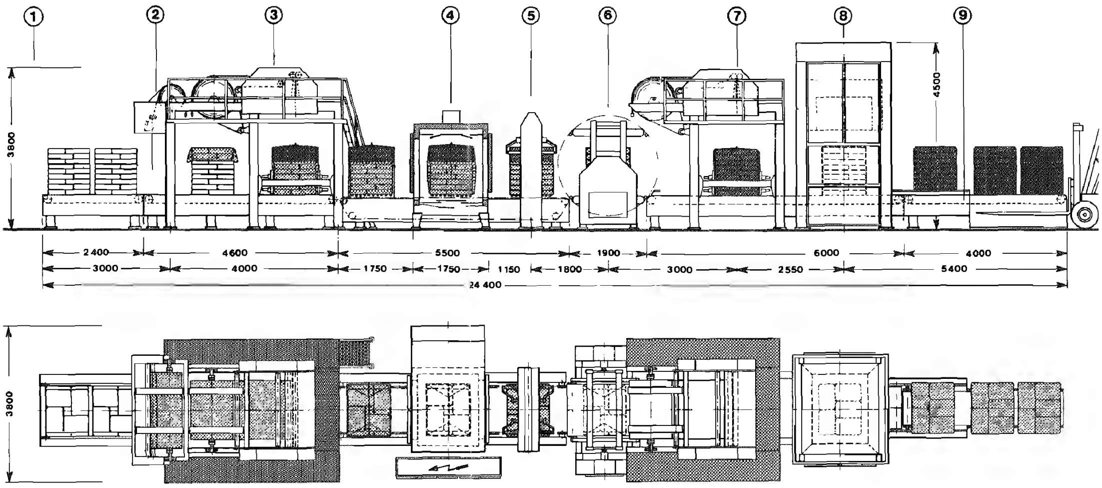
Fig. 23: "Big bag" filling terminal

means of a self-closing triangular device, which can serve also as a lifting attachment for transporting the filled sack over limited distances.
The re-usable sack likewise consists of plastic ribbon fabric with plastic sheet lining. It has an inlet and an outlet opening, enabling limited quantities of cement to be discharged to suit requirements. For transport and emptying, the sack can be suspended by means of straps. Though somewhat more expensive than the one-way sack, this type of "big bag" has the advantage that it can be used over and over again before becoming unserviceable.
Fig. 23 shows a "big bag" filling terminal.

5 Shrink wrapping

Especially in connection with shipments for export it is essential to protect cement packed in sacks, more particularly in the form of palletized unit loads, against moisture and also to ensure that these loads are well secured and stable so that the

sacks will not slip or topple down during handling and in transit. To achieve this, shrink-wrapping or stretch-wrapping of the whole unit load (the stack of sacks) including the pallet are techniques that have been applied for a number of years now. They suffer from some disadvantages, however, which are associated with the pallets themselves. Besides, the unit loads formed in this way are not completely protected against the weather on all sides, so that outdoor storage is possible only under suitable weather conditions.
A more recent development has been the "palletless" shrink-wrapping of stacks of sacks, thus eliminating the drawbacks of having to use pallets. The installations for this method of packaging produce large unit loads wrapped in shrunk-on thermoplastic film which are watertight, stackable and strong enough to withstand the buffeting they receive during handling and transport. As a rule, polyethylene film is used, which has the advantage of possessing considerable toughness and ductility, low water absorption, high resistance to chemical attack, good workability and lower price than other comparable types of film. For special purposes, polyethylene films containing so-called stabilizers are available, enabling them to withstand short-wave radiation, heat and other climatic influences.
Besides forming strong and conveniently storable and stackable unit loads, palletless shrink-wrapping has the advantage that the cement can be packed in ordinary two-ply paper sacks, whereas otherwise cement intended for shipment overseas usually has to be packed in five- or six-ply sacks.
The principle of this packaging method is as follows. First, the sacks are stacked in layers of five, with their pattern alternating from layer to layer so as to obtain interlocking. However, the last (top) layer consists of only three (or sometimes four) sacks which are so placed as to form a recess or ledge along each side of the stack. Later, when the stack has been shrink-wrapped and turned upside down for transport, the prongs of a fork-lift truck can be inserted into these recesses, which are now on the underside of the load. For the actual packaging operation there are several methods available, differing in details from one another. The individual layers of shrink-wrapping plastic film consist of flat sheets or of hoods formed from tubular film which envelop the stack and are heat-shrunk and bonded together. In this way the stack is finally enclosed in a strong, watertight, tough but elastic wrapping which is completely weatherproof and can withstand frequent handling operations by fork-lift trucks or cranes and the other forces to which it may be subjected during transport.
Figs 24a and b show shrink-wrapping lines using hoods of plastic film which are drawn down over the stack in opposite directions, after reversal of the stack. In the system shown in Fig. 24c the stack is first enclosed in a hood, reversed, and then covered with a top sheet which hangs down on all sides and is shrunk and bonded to the hood. The shrinking of the plastic film wrappings is accomplished by passing the wrapped stack through a shrink tunnel (continuous oven).
The first machine in any such shrink-wrap packaging line is an automatic palletizer which, for this purpose, must be able to complete the stack – composed of five-sack layers – with a final layer containing only three (or four) sacks.



① Palettierautomat Paletpac®	① Palletizing automat Paletpac®	① Automate de palettisation Paletpac®	① Autómata de paletización Paletpac®
② Deckfoliengeber	② Cover Film Placer	② Distributeur de feuille de couverture	② Distribuidor de lámine envesedore
③ Folienhaubenautomat	③ Film Hood Automat	③ Appareil automatique de pose de la housse	③ Equipo automatico de colocación de envese
④ Schrumpffolienerhitzer	④ Shrink Film Oven	④ Four de rétraction	④ Horno pera contrección de lámine
⑤ Kantenformer	⑤ Gusset Former	⑤ Formeur des coins	⑤ Modelador
⑥ Wendavorrichtung	⑥ Steck Turning Device	⑥ Dispositif de retournement	⑥ Dispositivo volteador
⑦ Deckfoliengeber	⑦ Cover Film Placer	⑦ Distributeur de feuille de couverture	⑦ Distribuidor de lámine recubridore
⑧ Schrumpffolienhaubenofen	⑧ Shrink Oven	⑧ Housseuse	⑧ Horno para la rétraccion de fundes de plestico
⑨ Förderanlagen	⑨ Conveyor	⑨ Transporteurs	⑨ Transportadores

Fig. 24a: Automatic shrink-wrapping lines for forming palletless unit loads composed of stacked sacks: konterpac process

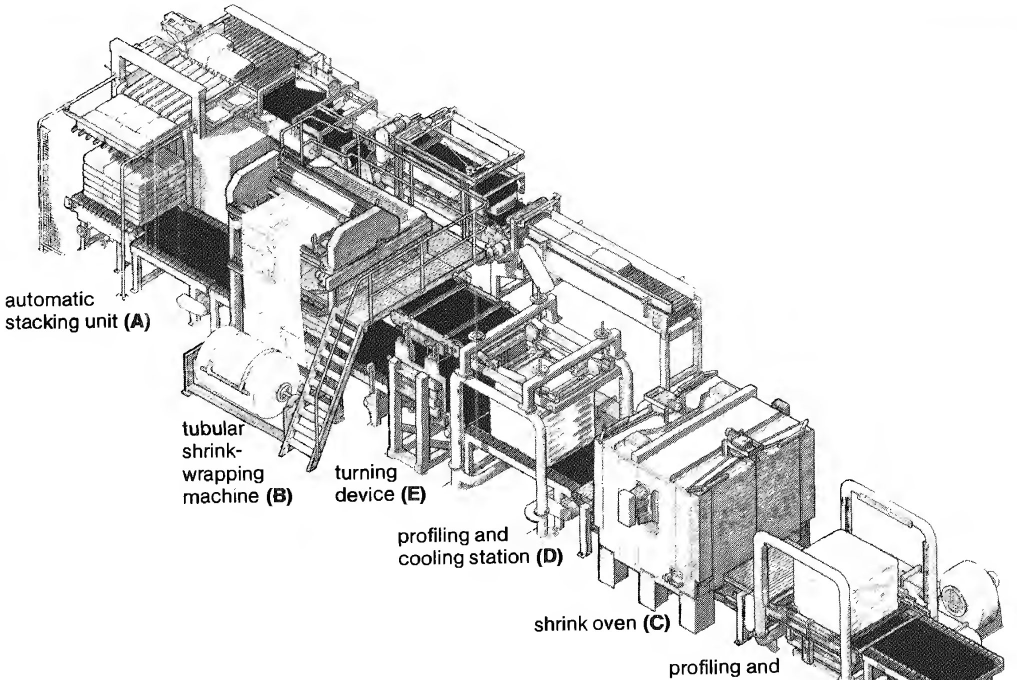
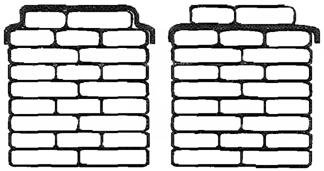
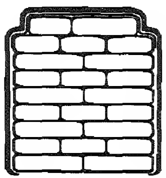


Fig. 24b: Automatic shrink-wrapping lines for forming palletless unit loads composed of stacked sacks: reverse hood shrink-wrapping method



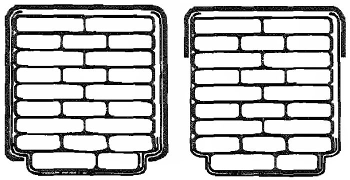
Phase 1

The base hood, or alternatively the intermediate hood is used for the reinforcement of the shrink hood of the main load areas. As a result of the application of thinner shrink film for the remaining surfaces material costs will be reduced



Phase 2

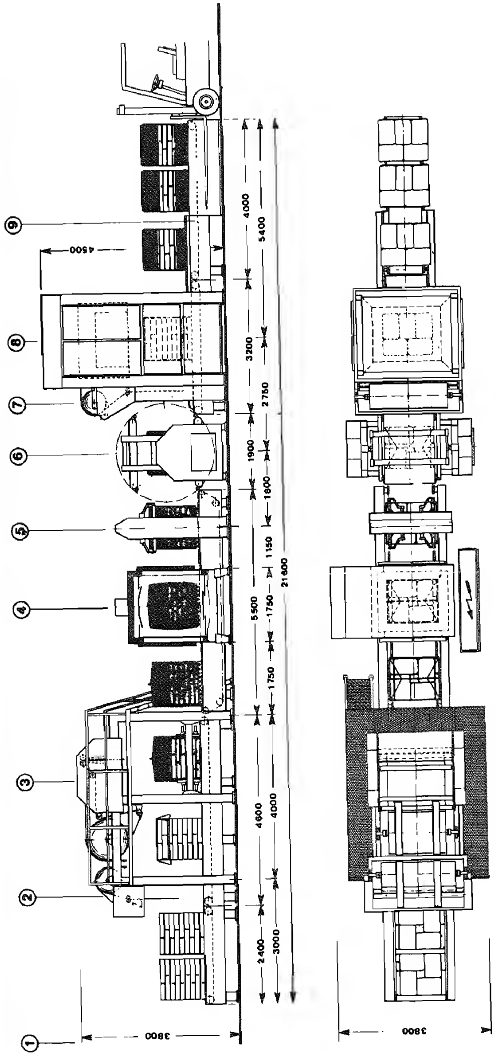
The inner hood covers the package entirely – this is the basic requirement for the mechanical stability of the palletless despatch unit and results, at the same time, in complete water-tight cover. Upon completion of the first shrinking process of base and inner hood the whole package is turned 180°



Phase 3

Depending on weight of palletless despatch unit, kind of bag, dimension and stability of different packages material savings are possible by infinitely variable adjustment of reverse hood (short and complete hood). During final shrinking process same is laminated to inner hood thus forming hermetically sealed watertight envelope

Fig. 24c: Automatic shrink wrapping



1 Palettierautomat Paletpac®
2 Deckfoliengeber
3 Folienhaubenautomat
4 Schrumpffolienerhitzer
5 Kantenformer
6 Wendevorrichtung
7 Deckfoliengeber
8 Schrumpffolienhaubenofen
9 Förderanlagen

1 Palletizing automat Paletpac®
2 Cover Film Placer
3 Film Hood Automat
4 Shrink Film Oven
5 Gusset Former
6 Stack Turning Device
7 Cover Film Placer
8 Shrink Oven
9 Conveyor

1 Automate de palettisation Paletpac®
2 Distributeur de feuille de couverture
3 Appareil automatique de pose de la housse
4 Four de rétraction
5 Formeur des coins
6 Dispositif de retournement
7 Distributeur de feuille de couverture
8 Housseuse
9 Transporteurs

1 Autómata de paletización Paletpac®
2 Distribuidor de lámina envasadora
3 Equipo automatico de colocacion de envase
4 Horno para contracción de lámina
5 Modelador
6 Dispositivo volteador
7 Distribuidor de lámina recubridora
8 Horno para la rétraccion de fundas de plastico
9 Transportadores

Fig. 24d: Automatic shrink-wrapping lines for forming palletless units loads composed of stacked sacks: flat film process

6 Automation of despatch procedures

In recent years cement producers and manufacturers of cement plants have been striving to develop and introduce methods, systems and forms of organization with the aid of which complex computer-controlled despatch facilities can co-ordinate and combine the movements of all products leaving the plant and also the arrival of certain materials coming into the plant (additives for cement production, any products returned from customers). This automation concept comprises a computer system for data acquisition, data storage, despatch operations control and despatch data ouput. The vehicle weighbridges and loading installations are also linked to the computer. On arrival at the cement works each vehicle driver is issued an identification badge. He inserts this into a badge reader and states his requirements. He is automatically instructed to proceed to a particular loading bay, where he himself carries out the loading operation, on completion of which he is automatically issued a delivery note. The despatch data for customer invoicing, financial accounting, etc. are fed into the commercial electronic data processing system of the cement plant [11 a].

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Acknowledgements for illustrations

Figs. 1–12 and 23 Haver and Boecker, Oelde/Westf., W. Germany
Figs. 13, 14, 15a, 16–24a, 24d Beumer Maschinenfabrik KG, Beckum/Westf., W. Germany
Figs. 15b and 24b, c: Möllers, Beckum/Westf., W. Germany

F. Handling and feeding systems – Continuous conveyors

By F. Mechtold

I. General introduction

As employed here, the term relates to material handling devices which run continuously. The material itself may be carried along in a continuous flow (e.g., on a belt conveyor) or in individual receptacles which may be very closely spaced (e.g., on a bucket conveyor) or farther apart (e.g., on a bucket elevator) or indeed some considerable distance apart and possibly detachable (e.g., on aerial ropeways or tramways).

In practice an optimum handling system in any given case may require a combination of two or more types of continuous conveyor, as is exemplified by the clinker handling system shown in Fig. 1. The arrangement illustrated here can be varied by using swing bucket elevators in lieu of the handling devices 5, 6 and 7, in which case the second bucket elevator 6 for lime and gypsum will also be omitted, because a swing bucket system can handle two or more different materials simultaneously and yet separately from one another. Further information on these various types of conveyor and elevator is given in the relevant sections of this chapter.

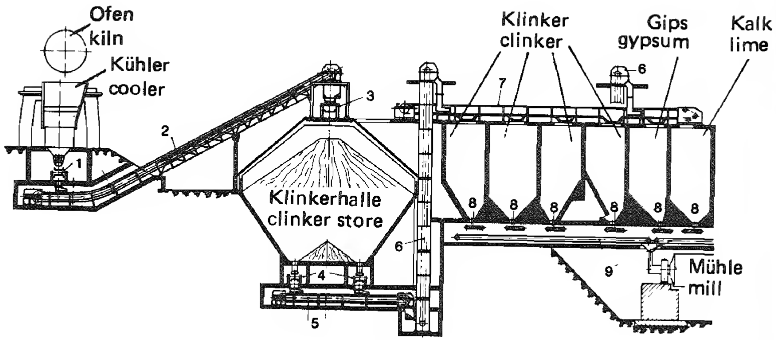


Fig. 1: Diagram of clinker handling system at a cement works

1 drag-chain or short-plate apron conveyor; 2 bucket conveyor or short-pan apron conveyor; 3 drag-plate apron conveyor for material distribution; 4 short-plate apron conveyor for extraction from hoppers; 5 short-pan apron conveyor for collecting; 6 bucket elevator; 7 drag-plate apron conveyor; 8 weigh belt feeder; 9 belt conveyor

In these the reader will find tables and/or diagrams giving essential information on handling capacities, drive power requirements, limiting values for conveying length, height, etc. These data are geared to practical needs, so that the desired information can be found quickly, without having to perform lengthy calculations. Obviously, it is not possible to give anything like an exhaustive treatment of the subject within the scope of this book. For further details the reader should consult specialized literature and the relevant standard specifications.
The notation and units employed here are as listed in Table 1.

II. Belt and band conveyors

1 Belt conveyors

Belt conveyors have been used for a great many years as handling devices for bulk materials and also for unit loads. They are the most widely used continuous conveyors because they are adaptable, versatile, reliable and economical. There has been much progress in the development of new and better types of belt in recent years, including the widespread use of synthetic fibre instead of cotton fabric for the carcass of the belt. The need for ever higher handling capacities has thus resulted in conveyor belts made with all-synthetic polyester or polyamide fabrics which are characterized by substantially higher tensile and impact strength and superior deformability in respect of stretch and troughing of the belt. Newly

Table 1: Notation used in formulas

B	mm	width of conveyor
h_3	mm	height of side walls of trough or casing
h_4	mm	height of transverse wall of casing
H	m	conveying height (ascending: positive; descending: negative)
J_M	t/h	mass flow
J_V	m^3/h	volume flow
K	–	reduction factor to allow for inclination of conveyor
L	m	distance between centres
P_H	kW	power consumed in raising the material
P_S	kW	power consumed in overcoming special frictional resistances
P_{mot}	kW	motor power rating
t	mm	chain pitch
v	m/s	conveying speed (= circumferential velocity in screw conveyor)
β	–	angle of repose of material being handled
δ	–	angle of inclination of conveyor
μ_w	–	coefficient of friction between material and wall or base
ϱ	t/m^3	bulk density of material being handled
φ	–	loading factor
λ	–	troughing angle of belt conveyor
D	m	external diameter of screw conveyor
d	m	shaft diameter of screw conveyor
s	m	pitch of screw conveyor
n	r.p.m.	speed of rotation

developed rubber mixes for the belt covers provide better wear resistance and, within certain limits, temperature resistance. In ambient temperatures above 50° C it is necessary to use special high-temperature belting, the best grades of which can, under short-term loading conditions, withstand temperatures up to 180°–200° C. A general drawback of belt operation at elevated temperatures is the accelerated ageing of the rubber. Thus, at 120° C the service life of the belt is halved. For this reason, various types of apron conveyor have largely superseded "rubber" belt conveyors for the handling of hot materials.
The best protection for the belt carcass, which is the actual pull-transmitting "structural" element of the belt, is provided by suitably thick covers, particularly on the upper or carrying face of the belt, their function being to protect it from damage by lumps of material falling onto it (cushioning effect), which might otherwise puncture or tear the fabric carcass, and from wear by abrasive action. The thickness of the cover should be at least 2 mm on the upper and at least 1 mm on the lower face. Table 2 gives approximate values for extra cover thickness (in addition to the 2 mm minimum requirement) on the upper face of belts for handling various types of material and for various types of loading onto the belt at the feed point.

The required minimum belt width depends on the following factors:
(a) the required handling rate;
(b) the maximum particle size of the material to be handled;
(c) the properties of the belt.

Table 2: Extra thicknesses for belt covers

feed point / properties of the material being handled		PARTICLE SIZE		
		fine	medium	coarse
		DENSITY		
		light	medium	heavy
		abrasiveness		
LOADING CONDITIONS	LOADING FREQUENCY	low	medium	severe
unfavourable	frequent	3–6	6–10	10–15
medium	medium	1–3	3–6	6–10
favourable	infrequent	0–1	1–3	3–6

Table 3: Relation between width of belt and particle size of the material to be conveyed

maximum edge length								
graded material	mm	100	150	200	300	400	500	600
ungraded material	mm	60	90	130	190	260	330	390
minimum belt width	mm	400	500	650	800	1000	1200	1400

Narrow belts, especially if they contain a high proportion of steel wire or other reinforcement, are more difficult to form into a transversely troughed shape than wide ones. In such circumstances the narrow belt will rest only with its edges on the side idler rollers and not be properly true-running. Although this problem can be eased by the use of very flexible belting fabric with specially supple weft threads, troughing angles of 30 degrees or more should be used only with fairly wide belts. The relation between minimum width and the particle size of the material to be handled is indicated in Table 3, where the belt widths standardized in Germany are given. Obviously there are certain economic limiting belt speeds, depending on the nature of the material to be handled by the conveyor. These are given in Table 4. There are standardized belt speeds: 0.84, 1.05, 1.31, 1.68, 2.09, 2.62 m/second.

Certain relationships between belt width and idler roller diameter and idler spacing should be conformed to in order to avoid subjecting the material being handled to a "rough ride" and especially also to keep the drive power input as low as reasonably possible. These data are indicated in Tables 5 and 6. The volume flow rates J_V (in m^3/hour) attainable with various belt widths and troughing angles of the belt, for a belt speed of 1 m/second, are given in Table 7, while reductions in handling rate due to upward slope of the belt are taken into account by means of the factors k in Table 8.

The angle of inclination δ should not exceed 15–18° if ordinary (plain) belting is used. The normal troughing angle is $\lambda = 20°$. The theoretical values given in Table 5 are, because of irregular feed of material to the belt, often exceeded by up to 50% in actual practice, so that a loading factor $\varphi = 0.7$ to take account of this should be applied.

The mass flow J_M (t/hour) can then be calculated as follows:

J_M (t/h) $= J_V$ (m^3/h) $\cdot$ v (m/s) $\cdot \rho$ (t/m^3) $\cdot \varphi$.

The required drive power of a belt can be approximately calculated with the aid of Table 9 and the following equation:

P_{motor} (kW) $= P_1 \cdot v + P_2 \cdot J_M \pm P_H + P_S$.

The power term P_1 for $v = 1$ m/second for various belt lengths is indicated in Table 9 and has to be multiplied by the actual speed v at which the belt is running. Similarly, the value P_2 must be multiplied by the mass flow rate (t/hour). The term P_H taking account of the belt inclination has the positive sign if the belt slopes upward in the conveying direction, and the negative sign if it slopes downward: $P_H = J_M \cdot H/367$.

The last term P_S represents the power losses due to ancillary equipment such as feeders, trippers, ploughs (scrapers), etc. For each additional device of this kind with which the belt conveyor is equipped, the following values should be added to the power consumption: 1 kW up to 650 mm belt width, 2 kW up to 1250 mm, 3–4 kW for larger widths. These values relate to a belt speed of 1 m/second. They should therefore be multiplied by the actual speed in m/second. If there are skirt plates to confine the material to the belt, an additional 0.1 kW power consumption per metre length of the conveyor should be allowed.

Where the conveyor has its transition from an upward inclined to a horizontal portion a convex curve occurs, which constitutes a kind of hump, as opposed to

Table 4: Economic limiting speeds in m/second

material to be conveyed	belt wear	belt width in mm 400	500	650	800	1000	1200
maximum particle size, graded or ungraded	medium	1.8	2.0	2.4	2.6	2.9	3.1
	severe, not sharp-edged	1.8	2.0	2.4	2.5	2.7	3.1
	severe, sharp-edged	1.3	1.8	2.1	2.4	2.6	2.8
0.5 maximum particle size, graded or ungraded	medium	1.9	2.0	2.6	3.2	3.5	3.7
	severe	1.8	2.0	2.5	3.0	3.3	3.5
graded, particle size 3 – 13 mm		2.1	2.5	3.2	3.7	4.1	4.3
fine, light, dry, dust-like,					1.0 – 1.3		
heavy					1.3 – 1.6		
friable, undesirable when comminuted					0.85 – 1.3		

Table 5: Diameter of idlers as a function of belt width

belt width	mm	400	500	650	800	1200	1600
light-duty type	mm	76	89	108	108	133	159
heavy-duty type	mm	108	108	133	133	159	193.7

Table 6: Average idler spacings

belt width mm	bulk density of material (t/m^3) 0.5	0.8	1.2	1.6	2.0	2.4
400	1.5	1.5	1.5	1.35	1.35	1.2
650	1.5	1.35	1.3	1.2	1 1	1.1
800	1.5	1.3	1.2	1.2	1.0	1.0
1200	1.2	1.2	1.0	1.0	1.0	0.9
1600	1.2	1.0	1.0	1.0	0.9	0.9

Table 7: Volume flow rates J_V in m^3/hour

belt width mm	handling rate in m^3/h for v = 1 m/sec. $\lambda = 0°$	$\lambda = 20°$	$\lambda = 30°$	$\lambda = 35°$	$\lambda = 40°$	$\lambda = 45°$
400	23	44	52	55	57	58
500	38	75	86	92	96	98
650	69	133	156	164	172	176
800	108	210	244	260	270	280
1000	173	335	394	415	435	445
1200	255	495	578	610	635	650
1400	351	680	798	840	875	900
1600	464	900	1050	1110	1160	1190
1800	592	1150	1345	1420	1480	1520
2000	735	1420	1670	1760	1840	1890

Table 8: Reduction factors k for various gradients

angle of inclination	δ	2°	6°	8°	10°	14°	18°	20°	22°
reduction factor k		1.0	0.98	0.97	0.95	0.91	0.85	0.81	0.76
angle of inclination	δ	24°	25°	28°	30°				
reduction factor k		0.71	0.68	0.61	0.56				

Table 9: Drive power requirements for belt conveyors

B \ L	5	8	10	12.5	16	20	25	32	40	50	63	80	100	125	160	200	250	320
power term P_1 for v = 1 m/sec. as a function of belt width and length																		
500	0.2	0.24	0.26	0.29	0.34	0.38	0.43	0.50	0.56	0.65	0.75	0.85	1.15	1.29	1.65	1.91	2.43	2.87
650	0.28	0.35	0.38	0.42	0.50	0.54	0.62	0.71	0.81	0.94	1.07	1.35	1.55	1.96	2.35	2.94	3.40	4.26
800	0.35	0.43	0.47	0.51	0.62	0.67	0.76	0.88	0.99	1.16	1.41	1.65	2.04	2.43	3.06	3.60	4.41	5.22
1000	0.54	0.66	0.75	0.81	0.93	1.04	1.18	1.35	1.57	1.78	2.17	2.55	3.09	3.64	4.59	5.35	6.54	7.72
1200	0.64	0.87	0.88	0.97	1.12	1.25	1.41	1.69	1.86	2.23	2.57	3.22	3.68	4.57	5.48	6.73	7.87	9.70
1400	0.77	0.95	1.05	1.18	1.35	1.49	1.70	1.96	2.24	2.74	3.14	3.88	4.45	5.51	6.62	8.09	9.49	11.76
power term P_2 as a function of belt length																		
	0.0027	0.0033	0.005	0.0041	0.0064	0.0052	0.0059	0.0068	0.0079	0.009	0.012	0.013	0.014	0.0163	0.0196	0.023	0.0265	0.0316

the concave curve at a transition from horizontal to an upward inclined portion. At a convex curve the edge zones of a troughed belt tend to be overstretched, whereas the opposite, i.e., overstretching at the centre of the belt, will occur at concave curves. As a rule a stretch of up to about 0.8% can be allowed. In connection with this, the angular deviation (in the vertical direction) from one idler to the next should not exceed a certain value, depending on the troughing angle of the belt, as indicated in Table 10. Where necessary, these permissible angles can be conformed to by closer spacing of the idlers. Table 11 gives limiting minimum values for the radii of belt curvature at concave and convex vertical curves. Further details on these matters are given in German Standard DIN 22101.

Table 10: Permissible angles of deviation at each idler set

troughing angle in °	20	25	30	35	40	45
max. deviation angle in °	3	2.5	2	2	1.5	1.5

Table 11: Minimum transition radii as a function of belt width

belt width	mm	500	650	800	1000	1200	1400	1600
R_{convex} = R_{crest}	$\lambda = 20°$	6.0	8.0	10.0	12.0	14.5	17.0	19.0
	$\lambda = 25°$	7.5	10.0	12.0	15.0	18.0	21.0	24.0
	$\lambda = 30°$	9.0	12.0	14.5	18.0	21.5	25.0	29.0
$R_{concave}$ = R_{valley}	m	60.0	75.0	90.0	120	150	170	190

According to information published in the literature, ordinary belt conveyors can be installed in horizontally curved alignments if the radius is not less than 1000 m There are as yet, however, very few examples of such installations actually built.

2 Steel band conveyors

This type of "belt" conveyor is equipped with a cold-rolled hardened steel band in lieu of a "rubber" belt and is used for special purposes. The thickness of the band is usually in the range of 1.0 to 1.5 mm. Because of the flat and smooth surface of the band, the material can very suitably be discharged by means of ploughs (or scrapers). Such conveyors are not very suitable for the handling of hot materials unless the band acquires a uniform temperature across its whole width; otherwise buckling is liable to occur in consequence of differential thermal expansion, causing serious trouble in the operation of the conveyor.

III. Bucket elevators

1 General explanation

This chapter will deal only with vertical elevators. Handling devices of comparable type for inclined conveying are included in the section on apron conveyors. Slow-speed bucket elevators (up to 0.7 m/second) discharge the material by gravity, i.e., it is simply tipped out of the buckets at the head sprocket or pulley. At higher speeds the centrifugal force plays a more significant part, and at speeds above 1.5 m/second it alone determines the discharge behaviour, i.e., the material is flung out of the buckets instead of merely falling out. See Fig. 2a. For efficient and complete emptying of the buckets, their shape, the design of the elevator head

assembly and the running speed must be correctly interadjusted. See Fig. 2b. The standard types of bucket elevator are indicated in DIN 151251, while bucket shapes are standardized in DIN 15231-37. Slow-speed bucket elevators with "internal" discharge are used more particularly for slightly sticky or caking materials, such as wet potash salts, or for friable materials which have to be handled "gently".

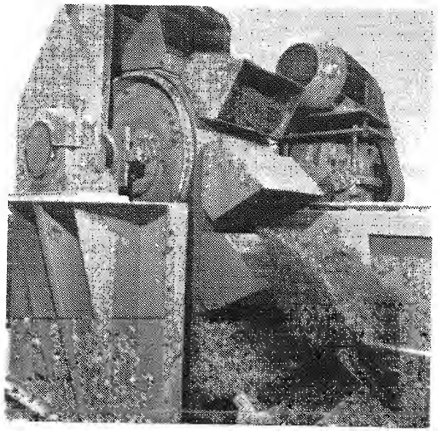


Fig. 2a: Centrifugal discharge of a high-speed belt bucket elevator

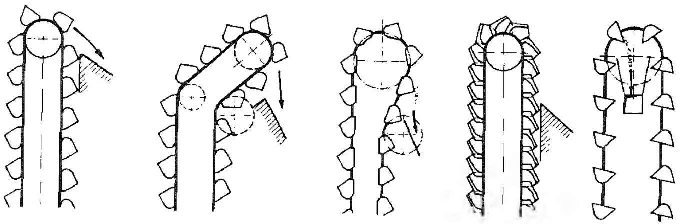


Fig. 2b: Various forms of bucket elevator

a high-speed elevator with centrifugal discharge; b low-speed bucket elevator with angled head and gravity discharge; c low-speed bucket elevator with snubbed return run; d low-speed elevator with continuously mounted buckets, each discharging over the preceding bucket; e low-speed bucket elevator with internal discharge

2 Belt bucket elevators

Cotton fabric belts as traction elements used to be employed for bucket elevators of the self-loading type – which scooped up the material by the digging action of the buckets – for the handling of light fine-grained materials (below 60 mm particle size). The desire to achieve greater elevating heights and to operate at higher temperatures led to the development of belts incorporating polyester and steel cable reinforcing elements. This has resulted in a general change in high-capacity bucket elevator engineering. Whereas chain bucket elevators are normally built for elevating heights of not more than 50 – 60 m, with steel cable belts it is possible to attain heights of up to 100 m. The limiting factor is now not so much the strength of the belt itself as that of the belt connectors for splicing the ends of the belt. A good deal of research on this aspect is still in progress. Figure 3a shows a commonly used belt connecting system. It has been found in practice that, under high tensile loading and with aging of the rubber covers to the belt, the steel wire cables are liable to be pulled out of the splice, resulting in parting of the ends of the belt. Steel cable belts provided with transverse reinforcement display more favourable behaviour in this respect. As an extra safeguard, however, the chain connecting device shown in Fig. 3a has been developed.

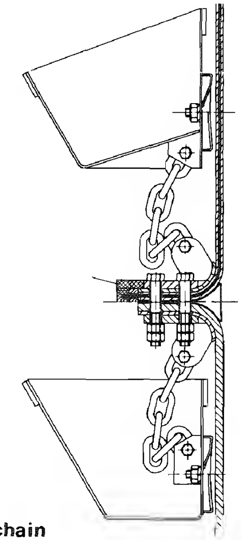
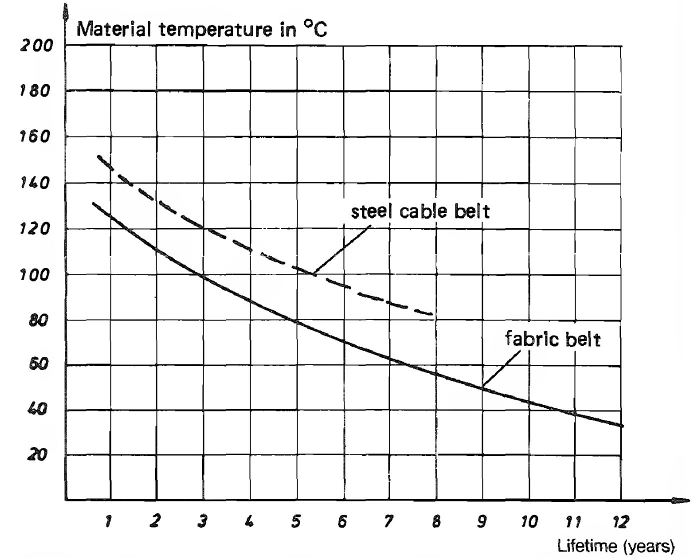


Fig. 3a: Clamped belt connection with safety chain

It is a well known fact that the rubber covers of the belts become brittle under the action of temperature in course of time. The fabric belts formerly employed, however, suffered from the particular disadvantage that elevated temperature (60–80°C) caused the carcass to age more rapidly than the covers. It was therefore extremely difficult to assess the internal condition of a belt. With the steel cable belt the situation is quite different. Here the action of elevated temperature will indeed cause embrittlement of the rubber covers, but will hardly affect the reinforcing cables. It is therefore possible, simply by visual inspection, to assess the condition of the belt and estimate its unexpired service liefe. As a rule, therefore, a steel cable belt will not fail suddenly; there is always enough time to procure a new replacement belt. This means, too, that the cost of keeping spares in stock is reduced. The relation between the service life of an elevator belt and the temperature of the material handled is shown in the accompanying diagram.



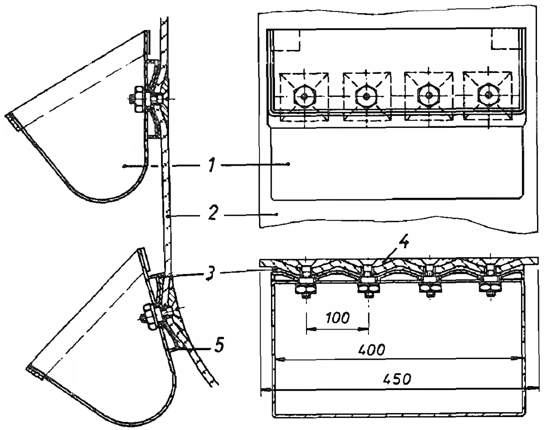
Relation between belt service life and temperature of the material handled

In general, therefore, elevated temperatures will shorten the service life of belts, especially fabric belts. Their effect on steel cable belts is less severe, which is a significant advantage because in material handling practice it is not always possible to keep within the design temperature limit.

Steel cable belts are also better able to withstand the action of foreign bodies. Damage to a single cable in the belt is not so critical as the punching or tearing of a hole in a fabric belt.

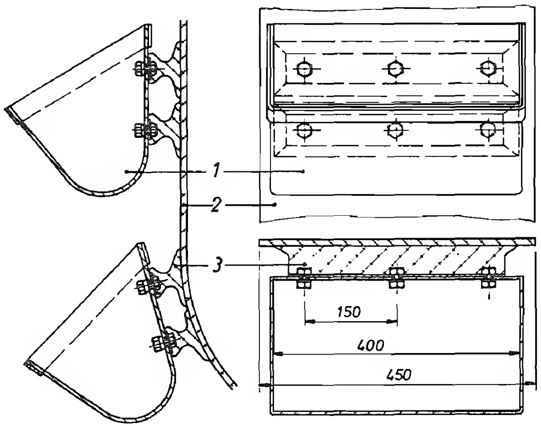
The buckets should be spaced as close together as possible on the belt in order to achieve satisfactory filling. Any material spilling out of the buckets has to be scooped up again, with the attendant disadvantages of extra power consumption, wear of the bucket edges, and heavier strain on the bucket attachments to the belt. The condition of the attachments and the belt itself should receive particular attention anyway. The reinforcing elements (cables) should be undamaged, otherwise they cannot safely be reckoned as transmitting the full design loads. Elevator belting has been developed in which there are certain longitudal zones in which no cables are present and in which the bucket fixing bolts can suitably be located. Fixing the buckets to the belt by simple bolting is normally confined to small installations with buckets up to 400 mm in width. To attain longer service life a layer of compressible material should be interposed between bucket and belt so as to ensure full-area contact at all times. This will prevent any fragments of hard material getting in between and becoming jammed there when the bucket passes round the end pulleys.

For the handling of material of up to 30 mm particle size the so-called segment fastening has proved very suitable (Fig. 3b), while the system shown in Fig. 3c is to be recommended for material with particles up to 60 mm in size. In this latter



1 = bucket
2 = belt
3 = intermediate ply of soft rubber
5 = segment strips

Fig. 3b: Segment fastening system for buckets



1 = bucket 2 = belt 3 = flexible mountings

Fig. 3c: Buckets fastened by means of flexible mountings

system each bucket is secured by bolting to two flexible special-profile rubber mountings which in turn are bonded to the belt. This method of bucket attachment is, however, suitable only for service at temperatures not exceeding 80° C.
Up to power ratings of 15 kW the usual method of belt bucket elevator drive is by means of gear-motors. For higher ratings the familiar drive systems – comprising the motor, starting clutch, reduction gear with non-reverse stop as individual units – are employed. An additionally fitted creep drive with overrunning clutch is convenient when the belt has to be inspected or repairs have to be carried out. The drive pulley of drum should preferably be provided with a 10 mm thick rubber surfacing to ensure good grip and power transmission. This surfacing should be "crowned" i.e., be convexly shaped in cross-section, for better belt guidance. Rubberizing the drive pulley in this way used to be very expensive, but has since been made simpler and cheaper by the use of rubber segments.
An accurately mounted – absolutely horizontal – tension take-up pulley helps to achieve correct running of the belt. The development of an automatically acting parallel guidance system calls for mention. It is suitable also for high-capacity chain bucket elevators and prevents the occurrence of slip between the chains and the non-toothed take-up wheels. In the case of belt bucket elevators the take-up pulleys are usually of the self-cleaning cage type with material-deflecting conical hugs, as shown in Fig. 4.
Further safety devices include: material level indicators in the loading hopper (at the foot of the elevator), a switch mounted on the tension shaft and responding to

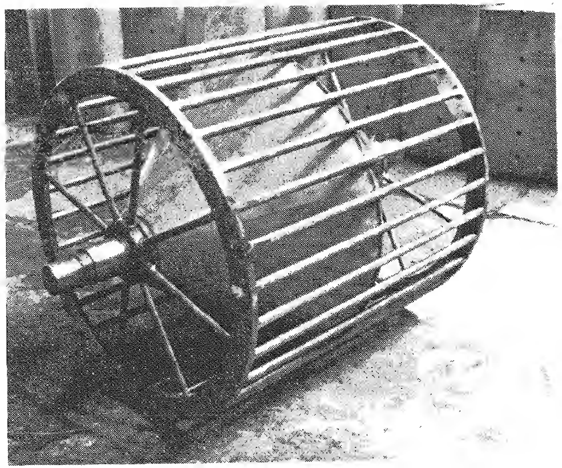


Fig. 4: Cage-type pulley with conical hub

belt slip, a true-run switch which responds to off-line running of the belt, and switches which stop the elevator if the buckets come into contact with the elevator casing at any point.
The belt bucket elevator is unsuitable for handling fairly hot materials – its main disadvantage. But it offers substantial advantages too: low wear, low drive power consumption, high mechanical efficiency, hardly and dynamic loading of the belt, and high handling capacity. The handling rates that can be attained are listed in Table 12a in the section marked "V". All the figures given there, including those within the dotted lines, are applicable to belt bucket elevators. Information on permissible bucket loading percentages is contained in Table 12b.

3 Chain bucket elevators

This is the only type of bucket elevator that can be used for the handling of hot materials. Besides bushed chains, round-link chains are also extensively used, their advantage being the smaller chain pitch giving quieter running on passing round the sprockets or chain wheels. For high-capacity bucket elevators it is necessary to use suitably heat-treated (quenched and tempered) steel chains of the round-link type in order to keep the amount of wear at the points of articulation within

acceptable limits. Another advantage of this open type of chain, especially for the handling of dry material consisting of angular particles, is that these will not attach themselves to the round articulation surfaces of the links and thus cause heavy wear. An elevator manufacturer has recently introduced a new round-link chain with larger articulation surfaces, so that the contact pressures are reduced. In this chain the individual round steel links are not interlinked in the usual way, but are mounted parallel side by side on pins with integral guide rollers. Outside the chain links these pins are provided with so-called drive rings with which the drive sprocket teeth engage (Fig. 5a).
These chains are particularly suitable for heavy loads, so that bucket elevators with large centre-to-centre distances and high handling rates with closely spaced buckets can be constructed with them. Similar results are, however, obtainable with bucket elevators having a central bushed chain. The original somewhat primitive pintle chains and bushed chains have, over the years, evolved into the heavy-duty long-lasting flat link chains based on German Standard DIN 8175 and having chain pitches of 160 or 180 mm (Fig. 5b).
The attainable handling rates depend on the chain running speed and on the bucket spacing. Closely spaced buckets also make for easier loading, thus reducing the scooping action involving heavy wear and power consumption. The material should be fed to the elevator at a uniform rate, and the discharge end of the feed chute should be substantially narrower than the buckets, while the chute should moreover not be steeply inclined. For further stepping up the handling capacity the so-called "W" bucket has been introduced, which encloses the chain on three sides and has a larger capacity. The attainable rates are listed in Table 12.

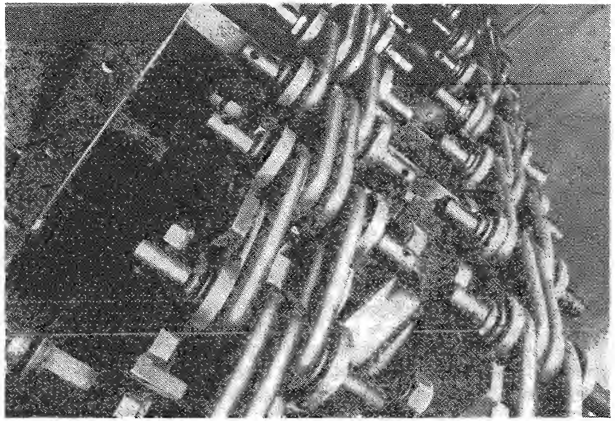


Fig. 5a: Round-link chain for high-capacity bucket elevators (special style of construction)

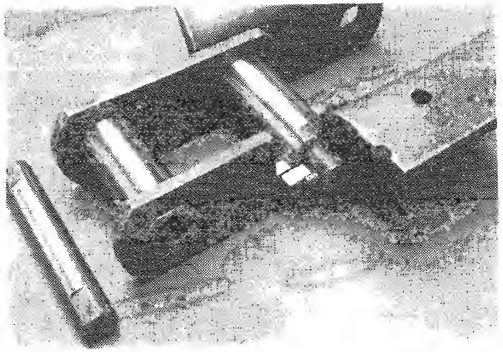


Fig. 5b: Construction of a heavy-duty flat link chain

The section "V" relates to buckets of the normal type. As already stated, all the figures in this section are valid for belt bucket elevators, but those within the dotted lines are not applicable to chain bucket elevators. The designations "VV" and "WW" relate to double bucket elevators comprising two normal single strands of buckets mounted side by side on the same drive shaft (Fig. 6).
The drive chain wheel at the head of the elevator is of three-piece segmental construction and has no teeth, force transmission being effected solely through friction. The great weight of the chain and buckets ensures that high frictional

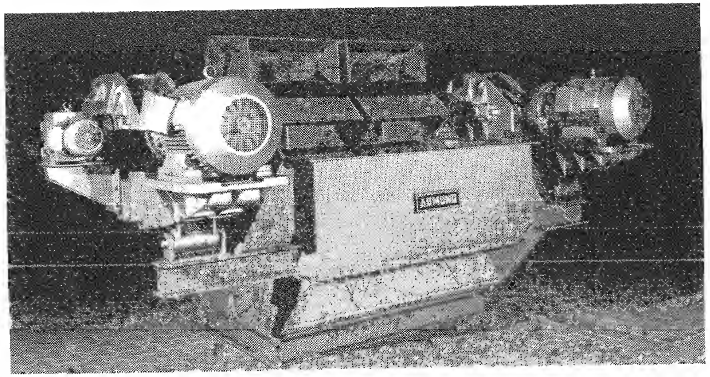


Fig. 6: Two single-strand bucket assembles on a common drive shaft

Table 12a: Capacity data for vertical bucket elevators

buckets shape	weight mm	over-hang mm	bucket spacing mm	bucket capacity dm^3	theoretical handling capacity m^3/hour, conveying speed m/sec. 1.05	1.16	1.29	1.42	1.58	1.73	1.95	2.16
V	250	200	320	4.6	**54**	60	67					
	280	200		5,1	60	**66**	74					
	315	200		5.6	66	**73**	81					
	355	250		9.6	113	**125**	139	154	171			
	400	250		10.8	128	**141**	**157**	173	192			
	450	250		12.1	143	158	**175**	194	216			
	500	250		13.5	160	176	**196**	216	240			
	560	250		15.1	178	197	**219**	242	269			
	630	250		17.0	200	222	**246**	272	303			
	710	250		19.2	227	251	**278**	307	342			
	800	250		21.6	255	282	**313**	346	385			
	900	250		24.6	290	317	352	389	433			
	1000	285	320	35.0						605	682	
	1250	320		55.7						**963**	1086	1203
	1400	320		62.4						**1079**	1216	1347
	1600	320		71.3						**1233**	1390	1540
VV	355	280	360	12.2	128	142	158	174	193			
	400	280		13.8	145	160	**178**	196	218			
	450	280		15.5	163	180	**200**	220	245			
	500	280		17.2	181	200	**222**	245	272			
	560	280		19.3	203	224	**249**	274	305			
	630	280		21.7	228	252	**280**	308	343			
	710	280		24.5	257	284	**316**	348	387			
	800	280		27.6	290	320	**356**	392	436			
	900	280		31.0	326	360	400	441	490			
W	400	355	360	13.0				185	205			
	450	355		15.2				216	240			
	500	355		17.5				248	**276**			
	560	355		20.2				287	**319**			
	630	355		23.3				331	**369**			
	710	355		26.8				380	**423**			
	800	355		30.8				438	**487**			
	900	355		35.4				503	560			
2 V	450	250	320	24.2			350	388	432			
	500	250		27.0			392	**432**	480			
	560	250		30.2			438	**485**	535			
	630	250		34.0			493	**544**	605			
	710	250		38.4			557	**614**	684			
	800	250		43.2			627	**690**	768			
	900	250		49.2			670	785	875			

Note:

The part of the table enclosed in dotted lines is valid only for belt bucket elevators. The capacities indicated are theoretical values for buckets filled flush and for continuous feed. Figures in heavy type are preferred standardized values. Practical operating data are obtained by allowing for the loading percentages (filling ratios) listed in the following table. The rate in t/hour is obtained by multiplying the value from the table by the bulk density of the material to be handled.

Table 12a: Capacity data for vertical bucket elevators

buckets					theoretical handling capacity m^3/hour							
shape	weight	over-hang	bucket spacing	bucket capacity	conveying speed m/sec.							
	mm	mm	mm	dm^3	1.05	1.16	1.29	1.42	1.58	1.73	1.95	2.16
2 VV	450	280	360	31.0			400	440	490			
	500	280		34.4			443	**488**	543			
	560	280		38.6			498	**548**	610			
	630	280		43.4			560	**616**	685			
	710	280		49.0			632	**695**	774			
	800	280		55.2			712	**784**	872			
	900	280		62.0			800	880	980			
2 W	500	355	360	35.0				496	552			
	560	355		40.4				574	638			
	630	355		46.6				662	**738**			
	710	355		53.6				760	**846**			
	800	355		61.6				876	**974**			
	900	355		70.8				1006	**1120**			

Table 12b: Permissible loading percentages

90 to 100%	per cent for pulverized or predominantly pulverized materials such as cement, raw meal and classifier tailings
60 to 90%	per cent for limestone, gypsum, coke, cement clinker, gravel and other materials up to 30 mm particle size
50 to 60%	per cent for aerated materials and materials of low specific gravity

forces can develop. With this system there is uniform wear of the wheel rim all round its circumference. The foot sprocket, with chain tensioning system for adjustment as the chain stretches or wears, is provided with coarse teeth for force transmission because here the dead weight of the chain and buckets is not available for developing high friction. The casing which encloses the elevator is usually made of steel plate and is a self-supporting structure. Alternatively, a concrete casing is sometimes used, which is constructed along with other parts of the building in which the elevator is installed. The internal width of the casing should be 300 mm more than the bucket width, while its dimension in the other direction will generally be 1600 or 1700 mm, depending on the chain wheel diameter.

DIN 22200 gives guidance on calculating the drive power requirements for bucket elevators. Here only a simplified method will be indicated, based on the fact that the term P_H representing the actual lifting power input is by far the dominant term in the equation (for comparison, see the equation for the inclined belt conveyor given earlier on), while the proportion required for overcoming frictional losses can be taken into account by means of a coefficient w:

$P_{motor} = w \cdot P_H = w \cdot J_M \cdot H/367$ (kW).

The following values may be adopted for the resistance coefficient w:

w = 1.2	for elevators whose buckets are fed, i.e., have no scooping or digging action to perform,
w = 1.7	for materials with low scooping resistance, e.g., cement,
w = 1.85	for materials with moderate scooping resistance, e.g., sand,
w = 2.1	for materials with high scooping resistance, e.g., crushed stone, cement clinker.

The problems associated with scooping resistance and material discharge conditions of bucket elevators have been the subject of research at the Technological University of Hanover, the results of which have been published in the literature.

4 Swing bucket elevators

The swing bucket elevator is especially advantageous in reducing environmental nuisance because it can convey materials both vertically and horizontally without necessitating transfer from one type of handling device to another, so that dust and

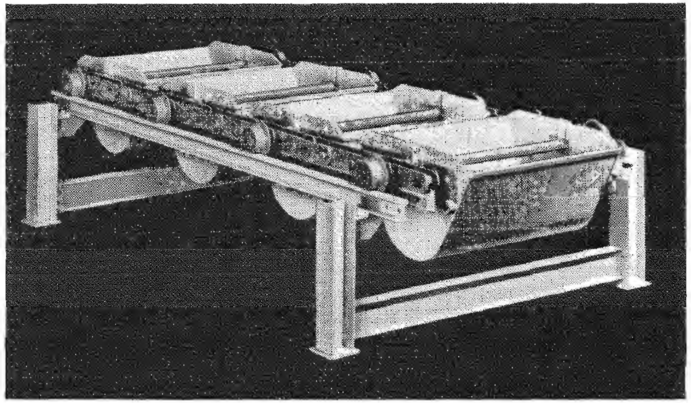


Fig. 7: Construction features of a modern swing bucket elevator

noise emission are kept to a minimum. By the use of wear-resistant materials for the bushed chains and running rollers it is possible to attain long service life, as is required in the cement industry. The large chain pitches (and bucket spacings) of 500, 750 or 1000 mm, as formerly employed, caused unquiet running (polygon effect of the chain wheels) and moreover required inconveniently large chain wheel assemblies. Nowadays, swing bucket elevators with a chain pitch of 250 mm are built, i.e., using standardized components of the same kind as those used for apron conveyors. See Fig. 7. The advantages associated with this development have resulted in revived interest in this type of elevator: quiet running, lower power consumption, interachangeability of standard parts with other material handling devices, continuous handling without material transfer points, possibility of simultaneously handling two or more different materials, discharge at various points, unlimited elevating height by installing intermediate drives.

The attainable handling rates in m^3/hour are indicated in Table 13. The rates in t/hour are obtained by multiplying these values by the bulk density of the material.

The elevating heights which can be attained without having to use intermediate drives depend mainly on the bulk density of the material to be handled and are given in Table 14.

In terms of space requirements for the installation it is to be noted that the construction depth (height) of the chain and buckets is 2550 mm, while the overall width is determined by the effective bucket width plus 1080 mm. In connection with requirements imposed by the filling and emptying operations the speed of the swing bucket elevator is restricted to a maximum of 0.45 m/second.

Table 13: Capacity data for swing bucket elevators

buckets			theoretical handling capacity m^3/hour					
overhang mm	width capacity mm	bucket dm^3	conveying speed m/sec.					
			0.1	0.25	0.30	0.35	0.40	0.45
770	600	135	49	122	146	**170**	194	219
	800	180	65	162	194	**227**	259	292
	1000	225	81	203	243	284	324	365
	1200	270	97	243	292	340	389	437
870	800	235	85	212	254	**296**	338	381
	1000	295	106	266	319	**372**	425	478
	1200	355	128	320	383	**447**	511	575
	1400	415	149	374	448	523	598	672
	1600	475	171	428	513	599	684	770

Preferred values in bold type. The handling capacity figures are based on water filling of the buckets. The actual maximum handling capacity should be reckoned as 70 – 80% of the theoretical capacity.

Table 14: Maximum elevating height as a function or bulk density

buckets		max. elevating height in m				
length a mm	width BW mm	bulk density in t/m^3				
		1.0	1.2	1.4	1.6	1.8
770	600	90	83	77	72	68
	800	76	70	65	60	56
	1000	66	60	55	51	48
	1200	59	53	48	44	41
870	800	72	66	61	56	53
	1000	62	56	52	48	45
	1200	54	50	45	42	39
	1400	49	44	40	37	34
	1600	44	40	36	33	31

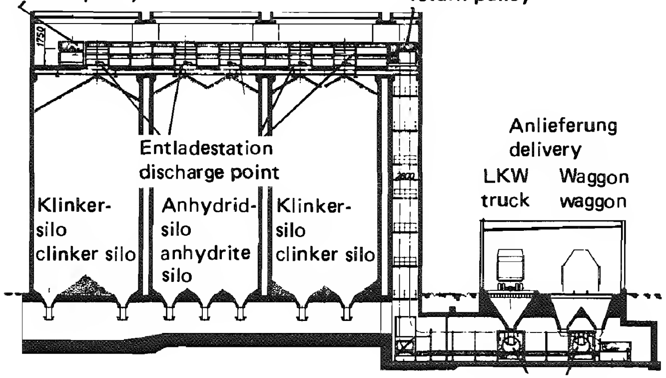


Fig. 8: Swing bucket elevator for the simultaneous handling of several materials

Filling the buckets requires particular attention so as to avoid spillage, since their edges do not overlap with one another. Various feed devices are available for the purpose, such as drum feeders, hopper chain feeders and reciprocating table feeders (Fig. 8). Alternatively, the buckets may be filled on an ascending inclined length of elevator, where they overlap with one another, as seen in vertical projection, so that any material spilled over the edge of a bucket will be caught in the next bucket. Emptying the buckets is done on special tipping devices comprising inclined rails forming ramps which are encountered by projections on the buckets. Such bucket emptying devices can be interposed into the conveying path and retracted by remote control.

For estimating the drive power consumption it is necessary to proceed step by step. For the vertical sections of the handling path the simplified calculation already presented for vertical bucket elevators should be applied, but now putting $w = 1.0$, as there is no scooping action at all. On the other hand, some appropriate allowance should be made for frictional losses at the feed and discharge devices if these are operated with drive power direct from the elevator itself. For the horizontal sections of the handling path the power consumption can be estimated in the same way as for an apron conveyor. An approximate value for the overall drive power, depending of course on the size of the swing bucket elevator concerned, is: $5-20\,\text{kW}/100\,\text{m} + J_M \cdot H/367$ (kW). The empty weight of the moving parts ranges between 120 and 350 kg/m.

IV. Chain conveyors

1 Flight conveyors

The simplest type of chain conveyor for bulk materials is the flight or scraper conveyor, which moves the material by pushing or scraping it along (Fig. 9). The scraper elements (flights) are attached to an endless chain, generally a bushed chain, which slides on a guide rail. In some cases there are twin chains with the flights mounted between them. In the simplest form of construction the material is conveyed in a trough without a bottom, this arrangement being more particularly used for filling long storage hoppers because it distributes the material very conveniently without requiring any special attention. Despite the drawback of heavy power consumption this may be the preferred type of conveyor for short distances. If the material is moved along in a steel trough with a bottom plate, the power demand is about 30% less. The material can be discharged at any intermediate point through bottom openings closable with slide gates.

The conveying speed varies, according to the type of material handled, from 0.2 m/second for coarse lumps to 0.8 m/second for finely granular material. Since the chains move in the material, wear at the articulations is inevitable. The contact pressure of the link connecting pins at these points within the chain should in general not exceed 4000 N/cm^2. Chains operating in highly abrasive materials should be so designed that this pressure is only about 1500 – 2000 N/cm^2.

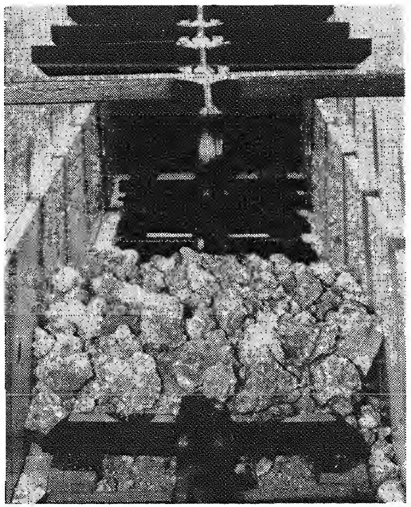


Fig. 9: A flight conveyor of robust construction

The handling capacity of a flight conveyor is determined by its width, the height (or depth) of the flights, their spacing, the loading (filling ratio) and the speed of the chain. The loading will depend on the internal friction of the material, which corresponds approximately to its angle of repose. The larger this angle, the higher is the column of material that will be carried along by the chain and flights. If the conveyor is heaped up higher with feed material, it will merely extract and carry away a layer of a certain depth from underneath. Thus there is no risk of overfilling, as in the case of a screw conveyor, for example. For this reason flight conveyors can extract materials directly from bins and hoppers. Theoretically attainable handling rates are indicated in Table 15. Actual values will generally be in the range of 80 to 90% of these.

The height of a flight is normally between 3 and 6 times the flight spacing. Its width will depend on the particle size of the material to be handled. With a twin chain

Table 15: Capacity data for flight conveyors

trough width b mm	shift height height h mm	theoretical handling capacity m^3/hour, conveying speed m/sec. 0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80
		– single-strand –							
200	200	14	27	41	55	68	82	96	109
250	200	17	35	52	69	86	104	121	138
	250	22	44	66	88	110	132	154	176
315	200	22	44	66	88	110	132	154	176
	300	33	67	100	134	167	201	234	268
400	300	42	84	126	168	211	253	295	337
	400	57	113	170	226	283	339	396	452
500	300	53	105	158	210	263	315	368	420
	400	71	141	212	282	353	423	494	564
	500	89	177	266	354	443	531	620	708
630	300	67	133	200	266	333	400	466	533
	400	89	179	268	357	446	536	625	714
	500	112	224	336	448	560	672	784	896
		– double-strand –							
800	500	141	283	424	566	707	849	990	1132
1000	500	177	355	532	710	887	1065	1242	1420
1250	500	222	444	666	888	1111	1333	1555	1777

conveyor handling unscreened material the width should be 3 to 4 times, and for screened material it should be 2 to 2.5 times, the maximum particle size. In the case of a single chain flight conveyor the corresponding values are 5 to 7 times, and 3 to 3.5 times, respectively. They are less favourable in this system because the single chain runs along the middle of the trough, so that the feed and discharge conditions are more difficult than in the twin chain system.

2 Continuous-flow conveyors

Because of its rather poor filling ratio, the handling capacity of a flight conveyor rapidly diminishes on upward slopes. Thus, for an inclination of 1 : 10 the capacity will decrease by about 25%. This drawback is substantially obviated in the continuous-flow conveyor, in which the bulk material moves along within a completely filled duct as a continuous core. It is a sophisticated form of flight conveyor with specially designed flights that move along entirely embedded within the material. Such machines can convey the material in any direction, including the vertical. There is, it is true, a certain amount of relative movement between the chain and the material it is carrying along with it, depending on the type of material, but as a rule this "slip" is under 15 per cent. On account of its method of moving the material, this system is sometimes referred to as an "en masse" conveyor. One of the earliest and most familiar examples of the type is the Redler conveyor.

Like the swing bucket elevator, the continuous-flow conveyor can therefore move the material vertically as well as horizontally. The swing bucket elevator is expensive in initial cost and takes up a considerable amount of space, but has the advantages of low wear and little maintenance. Also, it can handle coarse lumps of material. On the other hand, the continuous-flow conveyor is used only for pulverized, fine-grained or flaky materials, which are carried along in a totally enclosed duct, so that there is no dust nuisance. It is to be noted that these conveyors are quite unsuitable for dealing with sticky, corrosive or ungraded materials with hard constituents.

The conveying speed is between 0.1 and 0.4 m/second. Handling rates in m^3/hour should be taken from Table 15. Rates in t/hour are obtained by multiplying these values by the bulk density of the material concerned.

The height H listed in Table 15 refers to the height (or depth) of the flights in the flight conveyor and to that of the duct of the continuous-flow conveyor. As already indicated, the duct is completely filled, the material movement as a continuous "core" being based on the fact that the resistance developed by the specially shaped flights attached transversely to the chain is greater than the frictional resistance developed between the material and the walls of the duct. The material can be discharged at any desired point through a suitably positioned outlet opening provided with a gate or valve. Possible types and arrangements of such conveyors are illustrated in Fig. 10.

The power consumption of the chain conveyors (flight conveyors and continuous-flow conveyors) described here is dependent on many more factors than those which govern the power consumption of conveyors which carry the material, as

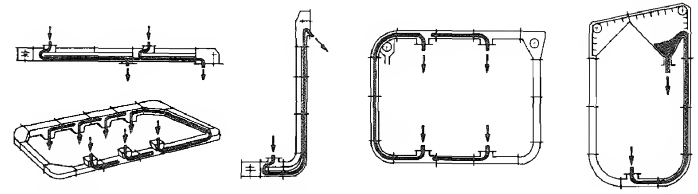


Fig. 10: Various types of continuous-flow conveyor

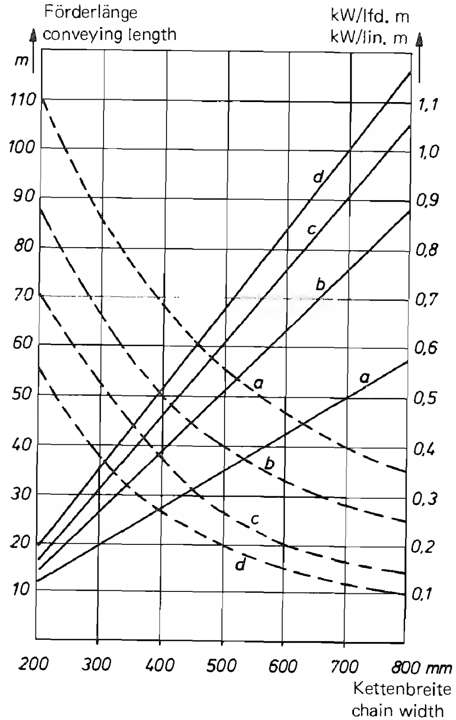


Fig. 11: Possible conveying lengths and drive power requirements of continuous-flow conveyors

opposed to pushing or scraping it along. Some approximate values for the power requirement per metre of conveying path, for various materials, are given in graph form in Fig. 11. The solid curve "a" relates to pulverized dry lignite (brown coal) with a bulk density of 0.5 t/m^3, curve "b" to coal (0.8 t/m^3), curve "c" to raw meal (1.25 t/m^3) and curve "d" to cement (1.4 t/m^3). The values in terms of kW/linear metre obtained from this diagram must be multiplied by the length of the conveyor. Besides, if the conveying operation involves raising the material to a certain height besides moving it horizontally, an additional power amount $P_H = J_M \cdot H/367$ must be taken into account. The dotted lines in the diagram indicate the maximum practicable centre-to-centre distances for the commonly employed chain sizes. For a material whose bulk density is intermediate between the values on which the curves in Fig. 11 are based it is permissible to interpolate between the curves or indeed use the curve corresponding most nearly to the bulk density in question. Continuous-flow conveyors are suitable for the handling of materials at temperatures ranging up to 200° C. Although the all-steel construction of such conveyors makes them fairly resistant to elevated temperatures, it is advisable to avoid higher ones, because the rate of wear on moving parts becomes much heavier, while distortion of the duct is liable to occur in consequence of local heat concentrations.

3 Apron conveyors

Apron conveyors of various kinds have come into widespread use in the cement and lime manufacturing industry, more particularly for the handling of hot and abrasive materials. Long service life and very modest maintenance requirements are the principal advantages. The same basic components in combination with different attachments for handling the material can be used for dealing with a variety of materials and circumstances. The drive and take-up assemblies, as well as the actual conveying path with its supporting frames and the two bushed chains with their carrying rollers (the spacing of which depends on the magnitude of the load per linear metre to be handled), are identical in the several apron conveyor types shown in Fig. 13. The actual material carrying elements ("aprons") bolted to the chain system may be short overlapping steel plates for normal material handling horizontally and on sloping paths of up to 18 degrees, buckle plates with convex upper surfaces for the extraction of sticky materials from hoppers and bunkers, short overlapping "trays" or "pans" for conveying on ascending slopes of up to 28 degrees, and "buckets" (deep pans) about 0.5 m in length for slopes of up to 60 degrees.

The size and strength of the chains conforming to DIN 8175 are determined with reference to the magnitude of the tensile force to be transmitted, while the spacing of the rollers depends on the weight of the conveyor and its load of material.

To ensure reliable operation even under very dusty conditions, the chain carrying rollers, of case-hardened drop-forged steel, mounted on ball bearings, can be fitted with dust-tight covers. The chain pitch is usually 160 mm for conveyors up to 1400 mm in width, and 250 mm for widths of up to 3000 mm.

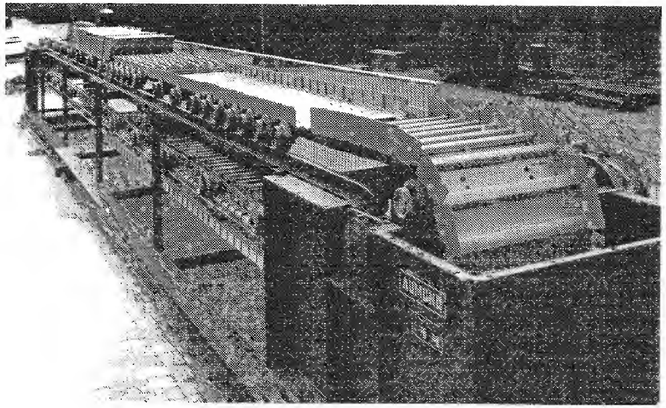


Fig. 12: Demonstration arrangement for various types of apron conveyor

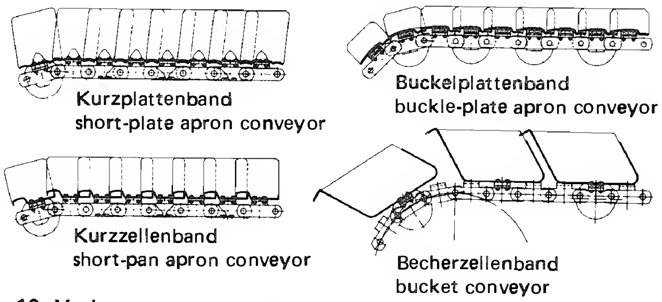


Fig. 13: Various apron conveyor types

With regard to the short-plate apron conveyor it is necessary, because of the close overlap of the plates, to have a minimum radius of 20 m on vertical curves in the conveying path. The "buckle-plate" (convex-plate) apron conveyor is similar in construction. It is especially suitable for sticky materials because its plates dispose themselves in a circular arc on passing round the chain wheel, enabling the adhering material to be scraped off easily.
Bucket conveyors can to some extent be regarded as inclined bucket elevators for slopes of up to 60 degrees. However, at such steep angles the filling ratio of the buckets is greatly reduced, so that for reasons of economy it is generally preferable

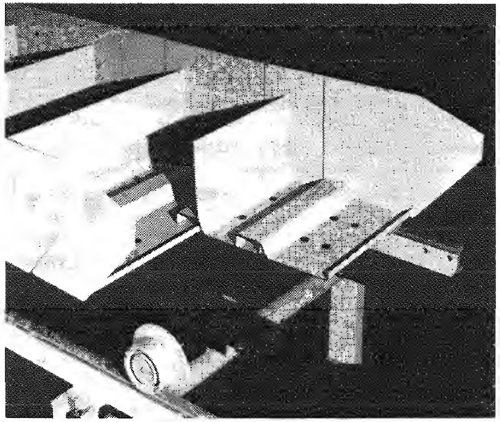


Fig. 14: Short apron conveyor for steep upward conveying

not exceed a slope of 45 degrees. Besides, it is then also more conveniently possible to arrange stairways beside the conveyor for access to carry out inspection or maintenance.
In recent years it has emerged that the handling rates attainable with a bucket conveyor can in many instances also be attained with a short-pan conveyor, if the latter is provided with partitions (transverse diaphragms) spaced at intervals of about 500 mm. This is a less expensive form of handling device, so that the short-pan conveyor can with some justification claim to be the universal conveyor of the future (Fig. 14). The handling rates that can be attained are indicated in Table 16. These values are applicable to conveying on ascending slopes of up to 28 degrees. The rate in t/hour is obtained by multiplying the values from the table by the bulk density of the material to be handled.
For steeper slopes (30 to 45 degrees) it is necessary to allow for reduced filling of the pans, as indicated by the factors given in Table 17. Preferred values for the side plate height are printed in heavy type in Table 16.
Bucket conveyors and short-pan apron conveyors have been in use for a good many years and have given ample evidence of their reliability and efficiency. They include conveyors with capacities of 300 – 500 t/hour and used for the raising of materials to heights of about 70 m.
For preliminary design purposes some further information on structural dimensions will now be given. Depending on chain wheel diameter, the overall height requirement is 1100 – 1400 mm for a short-pan conveyor and 1400 – 1800 mm for a bucket conveyor. The width occupied by the supporting frames is about 500 mm more than the net width of the aprons in all types of apron conveyor. On horizontal conveying paths it is possible to operate such conveyors of up to 1000 m length

Table 16: Capacity for short-pan apron conveyors

apron 250 Q		theoretical handling capacity m³/hour						
width b mm	height h mm	conveying speed m/sec. 0.10	0.15	0.20	0.25	0.30	0.35	0.40
400	**250**	24	36	48	60	73	85	97
	300	31	47	63	78	94	110	125
	350	38	58	77	96	115	134	154
	400	46	68	91	114	137	159	182
600	**250**	36	54	73	91	109	127	145
	300	47	70	94	117	141	164	188
	350	58	86	115	144	173	202	231
	400	68	102	137	171	205	239	273
800	250	48	73	97	121	145	169	193
	300	63	94	125	156	188	219	250
	350	77	115	154	192	231	269	307
	400	91	137	182	288	273	319	364
1000	250	60	91	121	151	181	211	242
	300	78	117	156	196	235	274	313
	350	96	144	192	240	288	336	384
	400	114	171	228	285	342	399	456
1200	250	73	109	145	181	218	254	290
	300	94	141	188	235	282	329	376
	350	115	173	231	288	346	403	461
	400	137	205	273	342	410	478	547
1400	250	85	127	169	211	254	296	338
	300	110	164	219	274	329	383	438
	350	134	202	269	336	403	471	538
	400	159	239	319	399	478	558	638
1600	250	97	145	193	242	290	338	387
	300	125	188	250	313	376	438	501
	350	154	231	307	384	461	538	615
	400	182	273	364	456	547	638	729
1800	250	109	163	218	272	326	381	435
	300	141	211	282	352	423	493	563
	350	173	259	346	432	519	605	692
	400	205	307	410	512	615	717	820

Table 16 (continued)

apron 250 Q		theoretical handling capacity m³/hour						
width b mm	height h mm	conveying speed m/sec. 0.10	0.15	0.20	0.25	0.30	0.35	0.40
2000	250	121	181	242	302	363	423	483
	300	156	235	313	391	469	548	626
	350	192	288	384	480	576	672	769
	400	288	342	456	569	683	797	911
2200	250	133	199	266	332	399	465	532
	300	172	258	344	430	516	602	689
	350	211	317	423	528	634	740	845
	400	251	376	501	626	752	877	1002
2400	250	145	218	290	363	435	508	580
	300	188	282	376	469	563	657	751
	350	231	346	461	576	692	807	922
	400	273	410	547	683	820	957	1093
2600	250	157	236	314	393	471	550	628
	300	203	305	407	509	610	712	814
	350	250	375	500	624	749	874	999
	400	296	444	592	740	888	1036	1184
2800	250	169	254	338	423	508	592	677
	300	219	329	438	548	657	767	876
	350	269	403	538	672	807	941	1076
	400	319	478	638	797	957	1116	1276
3000	250	181	272	363	453	544	634	725
	300	235	352	469	587	704	822	939
	350	288	432	576	720	865	1009	1153
	400	342	512	683	854	1025	1196	1367

Table 17: Loading factor for upward angle δ above 28°

side plate height mm	30°	35°	40°	45°
250	0.95	0.82	0.69	0.56
300	0.96	0.86	0.76	0.66
350	0.97	0.89	0.81	0.73
400	0.97	0.91	0.84	0.77

from a single drive station. The power requirement per 100 m length ranges from 4 to 15 kW, depending on the width of aprons. For ascending portions of the path it is of course necessary to make an appropriate allowance $P_H = J_M \cdot H/367$ (kW).
Normally, apron conveyors discharge the material only over the end, i. e., on passing round the head chain wheels, but there is a special system – the so-called drag-plate apron conveyor – which enables discharge of material to take place at any intermediate point along the conveying path. These conveyors are used more particularly for the handling of hot bulk materials. The length of each apron plate is equal to eight times the chain pitch, so that material consisting of lumps up to 500 mm in size can be handled. As shown in Fig. 15, the plates are pivotably mounted between the chains. The pivot is located somewhat off-centre in relation to the plate, one end of which rests on a roller. At a material discharge point the guide rails on which the rollers run are locally sloped down, so that the plates are tilted over and allow the material to slide off. This discharging operation can be remote-controlled by actuation of a swivelling section of rail. Alternatively, intermediate discharge can be achieved by means of throw-off carriages with which the material can be continuously deposited into longitudinal hoppers or onto longitudinal stockpiles and be suitably distributed. Good homogenization of the material is achieved at the same time.

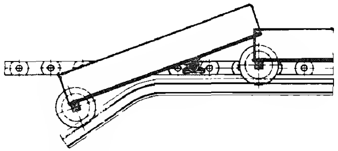


Fig. 15: Drag-plate apron conveyor

With the drag-plate apron conveyor the plates can be turned over at the head and tail ends of the conveyor, so that material can be carried on the return run of the chain as well. Various possibilities are thus available, as shown in Fig. 16. If the conveyor is used primarily as a means of cooling the material, the availability of the return run is advantageous in that it doubles the available distance travelled by the material while cooling. Any desired conveying distance can be attained by the use of intermediate drives. There are no problems in installing the conveyor, since the chains are outside the lateral edges of the plates and reliable engagement of the chain drive wheels with the chains is ensured by means of counter-rollers.
The drag-plate apron conveyor is usually equipped with side plates 150 mm or 200 mm in height. Recommended speeds are between 0.1 and 0.3 m/second. The attainable handling rates can be taken from Table 16 simply by halving the values for the side plate heights of 300 mm and 400 mm respectively, except that if decidedly coarse material is to be handled (300–500 mm particle size), a much smaller reduction need be applied: about 20%. For preliminary planning purposes an overall height of 2200 mm and a width equal to 800 mm more than the net width

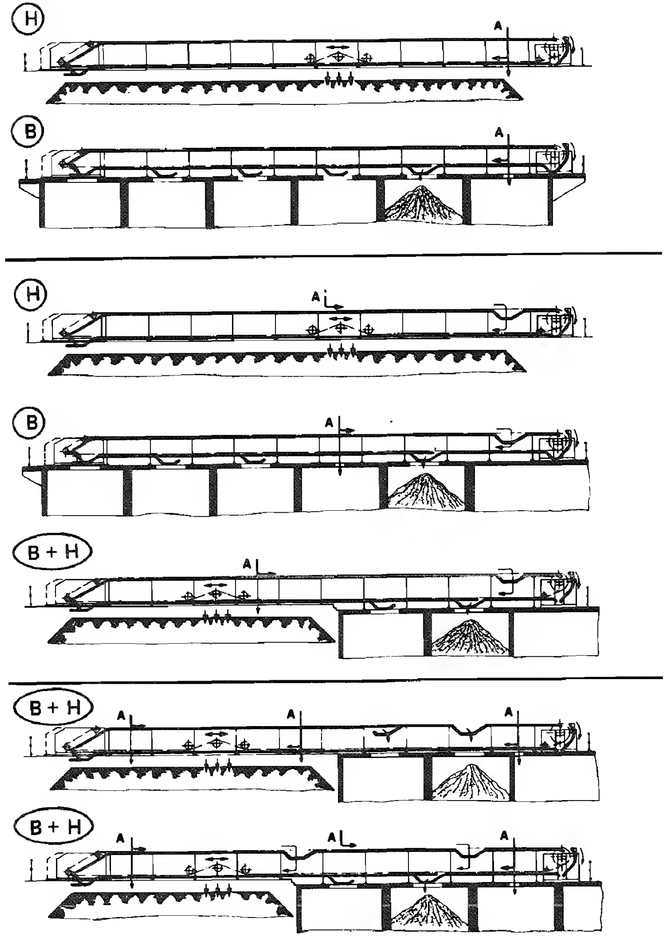


Fig. 16: Possibilities for using the drag-plate apron conveyor
H = stockpiling; B = bunker feeding; A = material feed

of the plates may be assumed. As with all conveyors of this general type, the drive power consumption depends on the width of the conveyor and, in this case, also on whether the return run will be used for conveying. Having regard to these considerations, the power consumption ranges between 5 and 25 kW/100 m conveying distance, plus an appropriate allowance for raising the material if there are inclined portions in the conveying path, as already indicated for the other types of conveyor.

V. Vibratory conveyors

The general term "vibratory (or vibrating) conveyors" comprises all material handling devices based on oscillating action which imparts so much acceleration to the material on the forward stroke that it continues to move forward by inertia forces and is not carried back by the return stroke. A general distinction is to be drawn between shaker conveyors (also known as jigging conveyors) and vibratory trough conveyors. What they have in common is that the conveyor oscillates, i.e., moves to and fro in the conveying direction. The difference is that the shaker conveyor operates at relatively low frequency and large amplitude, whereas the reverse is true of the vibratory trough conveyor. A more relevant distinction, however, lies in the fact that in the case of the vibratory trough there is, in addition to the horizontal motion, a vertical upward motional component with an acceleration exceeding that due to gravity; on the other hand, with the shaker conveyor there may likewise be a vertical component, but this is always smaller (in absolute value) than the gravitational acceleration.

The vibratory trough accelerates the material to such an extent that, when the forward movement of the trough is greatly retarded just before the end of the stroke, the material "becomes airborne" and travels in a parabolic trajectory until it falls onto the bottom of the trough, when it receives a fresh impulse, and so on. In this way it hops along, as it were. On the other hand, on the shaker conveyor the material is not thrown up into the air, but merely slides along, because on the return stroke of the conveyor trough the static friction between it and the material is temporarily cancelled.

Vibratory trough conveyors are used for handling granular and lumpy bulk materials in the horizontal or in a slightly inclined (ascending or descending) direction. They can be fed at any point along the conveying path, and material discharge can be effected at the end or at any intermediate point through closable bottom openings. Temperatures of the material to be handled ranging up to 700° C and particle sizes up to 600 mm no longer present any problems, the only limits being imposed by excessive moisture content or too high a degree of fineness of the material.

Open vibratory trough conveyors are used for all kinds of materials to be extracted from a bin or hopper, provided that they are not too fine-grained or liable to cause dust nuisance. The volumetric rate of discharge from a hopper is calculated as the product of three factors: width of the trough, depth of material in it, and conveying speed. The depth of the layer of material is determined by the shape of the hopper outlet and may be anything up to 600 mm (Fig. 17)

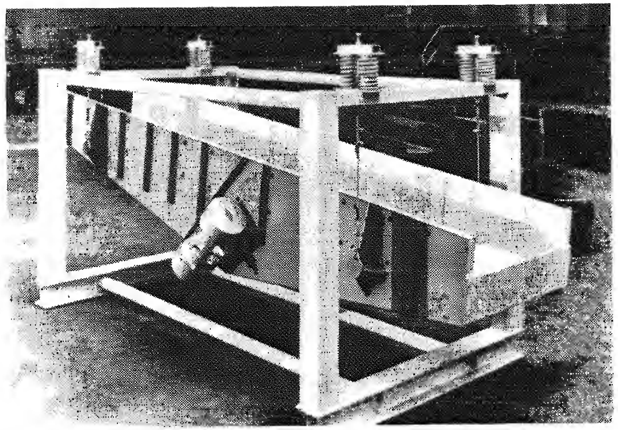


Fig. 17: Open vibrating trough conveyor with unbalanced-weight drive

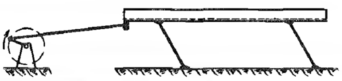
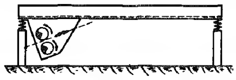
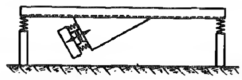
For materials tending to form dust the trough is provided with a fitted-on cover which participates in the vibratory motion or, alternatively, the trough may be enclosed in a non-vibrating outer casing. With these arrangements virtually the full cross-section of the trough can be utilized for conveying. If a closed duct (up to 500 mm diameter) instead of an open (covered or uncovered) trough is used, the loading must be reduced to only about 50% in order to avoid choking.

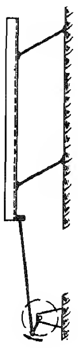
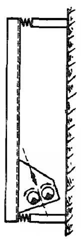
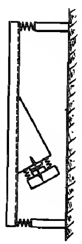
Shaker conveyors are usually driven by some form of crank mechanism. Unbalanced-weight drives or electromagnetic vibrators are used for vibratory troughs. A disadvantage of the crank mechanism and also of the unbalanced-weight drive is that there is a fairly long run-out time after switching off, so that the flow of material delivered by the conveyor does not stop at once. If rapid cut-off of the flow is required, e. g., for feeding a belt weigher or similar device, it will be necessary to apply counter-current braking. The electromagnetic vibrator is free from this drawback: it stops instantly when the current its switched off.

With crank drives the conveying speed can be varied by means of variable-speed gearboxes, variable-speed motors or induction couplings. With unbalanced-weight drives it is indeed possible to control the speed in large increments by using pole-changing motors or by applying frequency conversion control, which is very expensive, however. The simplest system for electromagnetic drives is thyristor phase-angle control, making this type of drive very suitable for extraction of materials from hoppers and bins under circumstances requiring changes in material handling rate while the conveyor is running.

As appears from what has so far been said about vibratory conveyors, the type of drive and its method of control constitute the most significant distinctive features of each system. The mode of operation of each type is more particularly determined

Table 18: Comparison of the principal data and characteristics of vibratory conveyors with crank, unbalanced-weight and electromagnetic drives

system	conveyor with crank drive	conveyor with unbalanced-weight drive	conveyor with electro-magnetic drive
			
frequency Hz	5–15–(25)	10–25–(50)	50; 100
amplitude mm	6–30	1–10	0,1–2
actuation angle degrees	25–35	20–30	20–30
downward slope degrees	0–5	0–15	0–25
maximum upward slope degrees	5–10	10–12	12–14
conveying speed m/s	0.3–0.7	0.05–0.4	0.01–0.15
length of trough m	2–20 special types: <50 m	0.5–10 special types: 30–50 m	0.1–5 special types: < 10 m
material handled	granular material, pulverized material, slightly caking material (conditionally),	granular, material pulverized material (conditionally), slightly caking material (conditionally), unit loads (conditionally)	granular material unit loads
purpose	conveying	conveying, extracting from bins, etc., feeding	conveying, extracting from bins, etc., feeding proportioning
trough wear (per kg of material handled)	extremely little	very little	little
gentleness of handling	little	moderate	good
drive	Crank or eccentric drive through vee-belts or variable speed gears by static motor with constant or variable speed.	One unbalance motor with undirected centrifugal force. One unbalance motor with directed centrifugal force. Two synchronously counter-rotating unbalance motors with directed resultant centrifugal force. Static motor with constant or variable speed and vee-belt or flexible shafts driving an unbalanced pulley or two synchronously counter-rotating unbalanced pulleys geared together.	Electromagnetic vibrator comprising an armature fixed to the trough and a spring-mounted mass (supported on preloaded leaf or coil springs). The electromagnet is normally energized through a half-wave rectifier for producing an operating frequency of 50 Hz.
drive power in kW for 10 m length and 10 t/hour capacity	0.6 kW	0.8 kW	0.7 kW

system	conveyor with crank drive	conveyor with un-balanced-weight drive	conveyor with electro-magnetic drive
			
support	By guiding leaf springs on foundation or countervibrating frame. By guiding leaf springs and additional energy-storing springs on counter-vibrating frame. By shear rubber springs on counter-vibrating frame.	By flexible bolts or rubber springs on foundation. By flexible bolts, rubber springs or guiding leaf springs, possibly with additional stiff energy-storing springs, on counter-vibrating frame. Suspension mounting possible.	By flexible bolts or rubber springs on foundation. By guiding leaf springs on counter-vibrating frame. Suspension mounting possible.
controllability of handling rate	By frequency control of the drive with the aid of a device for regulating the depth of the layer while running. By varying the crank throw when stopped.	By frequency control of the drive or with the aid of a device for regulating the depth of the layer while running. By adjusting the unbalanced weight when stopped.	By changing the magnet armature voltage with the aid of a series resistor, transformer or transductor while running.
example	Merz trough	Schenck trough	AEG vibratory trough

by the working frequency employed. To assist the works planner in selecting the most suitable vibratory conveyor for a given set of duties, Professor K. H. Wehmeier has prepared a systematic classification of the various types of vibratory conveyor and their properties (Table 18).

With downward inclined conveying paths the capacity of vibratory trough conveyors can be considerably increased, namely, by 3–6% per degree of slope in short, and by 2–3% per degree of slope in long conveyors. It should be borne in mind, however, that trough wear increases too. Hence it is preferable not to exceed an angle of 10–15 degrees.

Conversely, on an ascending path the capacity of the vibratory trough conveyor decreases by 2–3% per degree of slope. Under such circumstances, more particularly if the material to be handled has low permeability to air, voids are liable to be formed within the material and between it and the bottom of the trough. The air trapped in these voids adversely affects conveying efficiency, though considerable improvement can be achieved by appropriately interadjusting the amplitude and frequency and by reducing the depth of the layer of material in the trough.

In general, the unbalanced-weight conveyor represents the simplest form of vibratory conveyor. Any subsequent changes in the oscillating mass due to wear or accretions of material or indeed due to the addition of wearing plates in the conveying trough have no effect on operating behaviour (Fig. 18).

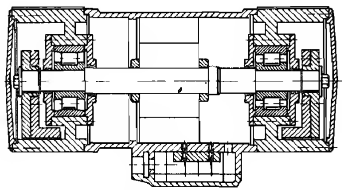
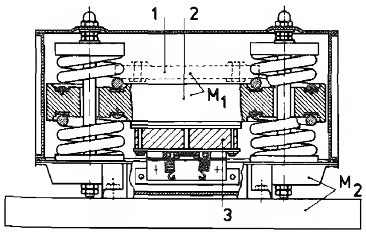


Fig. 18: Section through an unbalanced-weight motor



1 auxiliary mass
2 main oscillating mass
3 magnetic unit
M_1 oscillating mass
M_2 driven equipment

Fig. 19: Electromagnetic vibrator

The electromagnetically actuated vibratory trough conveyor is considerably more sensitive to changes in the oscillating mass, but is nevertheless the preferred type in cases where control of the handling rate during operation is required. See Fig. 19. Because of its relatively low conveying speed, this type of conveyor is wider and heavier. Whereas small electromagnetic vibratory trough conveyors (up to about 60 m^3/hour capacity) are generally cheaper than comparable unbalanced-weight trough conveyors, with higher capacities and more robustly constructed types the cost situation is reversed. Crank-actuated shaker conveyors can, in terms of cost, compete with the other types of vibratory conveyor only where very high handling capacities are required.

VI. Screw conveyors

Screw conveyors (also known as spiral or worm conveyors) are used for the handling of granular or powdered bulk materials on horizontal or slightly inclined (ascending or descending) conveying paths. They are among the oldest types of mechanical handling appliances. The material is pushed along in a trough, so that in this respect the conveying action is similar to that of a flight conveyor. Because of the simultaneous relative motion of the conveying element, i.e., the helix or screw (also referred to as the spiral or flight), there is not only the friction of the material against the sides of the trough to be overcome, but also the friction against the helix itself. Hence the power consumption is higher than that of a flight conveyor of equal capacity. The direction of conveying is determined by the direction of rotation of the shaft (clockwise or anti-clockwise) and/or the direction of the helix itself (right-hand or left-hand). For a given direction of rotation of the shaft it is possible to make different sections of the same conveyor move the material in opposite directions by providing such sections with a right-hand and a left-hand helix respectively. In this way, materials fed in at two different points can be brought together or, alternatively, material fed in at an intermediate point can be moved towards the two ends of the conveyor. In general, a screw conveyor can be fed with material at any point, and discharge can take place at the open end of the trough and/or one or more intermediate outlets, which may be provided with gates to open and close them as required. To avoid blockages, the outlet openings should be of the same cross-sectional area as the screw itself.
The handling capacity of the screw conveyor is determined by the diameter of the helix, its pitch, its rotational speed, the loading (filling ratio) and the nature of the material to be handled. This last-mentioned parameter is very important and constitutes a basis of subdivision into three classes:

(a) Light, non-abrasive and free-flowing materials such as flour, grain, dry pulverized coal. With these materials it is possible to operate at high conveying speeds and a loading of up to 45%.
(b) Fine-grained or small-sized materials which are not quite free-flowing, such as coal, coarse salt, etc. For these a loading $\varphi = 30\%$ must be assumed.
(c) Heavily abrasive, tough materials containing hard lumps and with poor flow properties, such as ash, sand, clinker and cement. For these: $\varphi = 15\%$.

The advantage of screw conveyors is their compact form of construction. Also, they are very suitable for the handling of dusty, toxic, explosive or evil-smelling materials, because the trough or duct can be made dust-tight or gas-tight and resistant to internal or external pressure. The constructional features are simple, lengths of up to 40 m are possible with a single conveyor, and ascending paths (usually up to 45 degrees) present no problem, while special forms of screw elevator can be used for vertical transport of bulk materials. Drive power is applied at one end or at both ends of the screw shaft.
Disadvantages are the large frictional losses and heavy wear, with high power requirement. Also, screw conveyors are unsuitable for dealing with hard and tough materials which are likely to cause jamming. Intermediate bearings for the shaft are especially unfavourable from this point of view. As a general principle, such bearings should always be suspended, i.e., attached to the cover of the conveyor trough.
Screw conveyors are very versatile and can be used for feeding and proportioning, emptying of bins or hoppers, mixing, etc. If the conveyor is heated or cooled, it can be used for carrying out chemical processes during the material handling operation. As a protection against attack by aggressive chemical agents the conveyor and its internal parts may be rubber-lined or be made of stainless steel or plastic. Special types of screw conveyor include those with a helix that decreases in pitch or tapers in the conveying direction, so that the material being handled is compacted, and those with a helix that increases in pitch and thus loosens the material. Besides the normal helix, various other forms of flight are available for particular duties: ribbon helix, paddle flights, etc. Also, there are double-flight conveyors which achieve smoother flow than single-flight types. See Fig. 20.

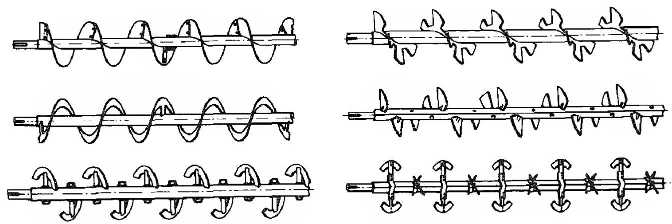


Fig. 20: Various forms of screw conveyor flights

Data on power consumption, handling capacity and dimensions of normal screw conveyors are given in Table 19.
Rotational speeds for the screw shaft range from 16 to 140 r.p.m., but are normally within the range of 50 to 100 r.p.m. The trend is towards lower speeds and larger diameters in order to reduce frictional losses.

Table 19: Power requirements and handling capacities of horizontal screw conveyors
$s = 0.75 \cdot D$; $\varphi = 0.2$; $\delta = 1.5\,t/m^3$; $v = 1\,m/s$

D/d	n	s	m^3/h	t/h	P_{motor} in kW for a screw length of m					
					5	10	15	20	25	30
200/55	95	0.150	4.96	7.44	0.46	0.63	0.79	0.93	1.07	1.20
250/55	77	0.188	8.16	12.24	0.79	1.05	1.29	1.52	1.75	1.97
300/60	64	0.225	11.75	17.63	1.13	1.51	1.85	2.21	2.52	2.83
350/60	55	0.263	16.14	24.21	1.56	2.08	2.55	3.02	3.46	3.90
400/75	48	0.300	20.91	31.37	2.02	2.68	3.30	3.90	4.49	5.04
500/90	38	0.375	32.50	48.75	3.13	4.16	5.12	6.07	6.97	7.84
600/100	32	0.450	47.52	71.28	4.60	6.08	7.51	8.91	10.19	11.45
700/110	27	0.525	63.78	95.67	6.88	8.17	10.08	11.92	13.69	15.31
800/125	24	0.600	84.67	127.00	8.24	10.86	13.36	15.82	18.18	20.31

Pitch of helix: $s = 0.5-0.8\,D$.
Shaft diameter: $d = 0.3\,D$ for small, $0.15\,D$ for large helices.
Spacing of intermediate bearings: 2.5–3.0 m.
Permissible loading (filling ratio): $\varphi = 0.15$ to 0.45.
circumferential velocity $v = 1.0-1.5$ m/s.
The outside diameter of the helix can be calculated from:

$$D = 2 \cdot \sqrt{J_V/\pi \cdot s \cdot n \cdot 60 \cdot \varphi \cdot k},$$

where k is a factor depending on the length of the helix and decreasing almost linearly from a value of 1.0 for very short conveyors to 0.82 for a length of 40 m; J_V is the volume flow rate (m^3/hour).

The permissible inclination of the screw conveyor is limited by the pitch of the helix and the cohesiveness of the material to be handled. Normally, a maximum of 15 degrees is to be regarded as the economic limit. Handling capacity decreases by about 2% for each degree of slope.

The power required to drive screw conveyors depends on the length of the conveying path, the handling rate (throughput), the bulk density of the material, the pitch of the helix, and the resistances due to the nature of the material to be handled. The values listed in Table 19 are based on a pitch $s = 0.75\,D$, loading $\varphi = 0.2$, bulk density $\delta = 1.5\,t/m^3$ and circumferential velocity of the helix $v = 1.0$ m/second. The figures under P_{motor} indicate the power consumption for various horizontal conveying distances of up to 30 m.

A variant of the conventional screw conveyor is the type which consists of an axially rotating tube with helical flights attached to the inside, so that there are no independently moving parts in the tube, which is supported on rollers and driven from the outside. With this system the frictional losses and therefore the power consumption are less than in the conventional system, and there is also less risk of blockage or obstruction. On the other hand, the material is tumbled around in the tube, so that friable particles are more liable to disintegrate. Another drawback is that the tubular screw conveyor can be charged and discharged only at the ends; intermediate outlets are not practicable.

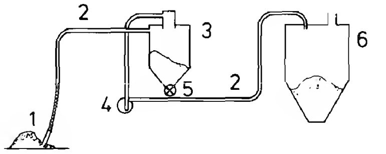
VII. Pneumatic conveyors

The conveying of solids in a gaseous carrying medium, usually air, has acquired considerable importance in industry in the last fifty years and is still gaining ground. A special advantage of pneumatic handling and conveying is that, in conjunction with the actual conveying operation, various chemical and physical processes involving the interaction of the gas and the material may advantageously be performed (catalytic processes, blending, drying, classifying, etc.). Other advantages are constructional simplicity, good adaptability, the complete absence of moving parts along the conveying path, freedom from pollution or dust nuisance, very modest maintenance requirements, weather resistance and, not least important, the suitability for automated operation.

A disadvantage of pneumatic conveying in general is the high power consumption and, with certain materials, the heavy abrasive wear of the conveying pipeline and of the material itself. With finely pulverized combustible materials there may moreover be explosion hazards under certain circumstances. Also, with certain materials there is a risk of blockage of the pipeline, especially at bends.

In a pneumatic conveying system the material to be conveyed is introduced into a stream of air by means of a feed device. The particles of material are carried along in the air through a pipeline. Depending on the mode of action, a distinction is drawn between pressure systems and suction systems. The latter type is preferred in cases where conveying has to take place from several feed points to one discharge point or where the feed point has to be mobile. However, such suction installations are confined to relatively short conveying distances and low loading densities of the air with the material to be handled, as the maximum possible conveying pressure is limited to the atmospheric pressure.

Pressure systems are employed where materials have to be conveyed over longer distances and with higher loading densities of the air, especially if the material has to be delivered from one stationary feed point to several discharge points. Combined installations are also used, in which the ease of material intake of the suction system is available in conjunction with the advantages of the pressure system (Fig. 21).



1 suction nozzle;
2 conveying pipline;
3 separator;
4 fan,
5 rotary valve;
6 hopper

Fig. 21: Combined suction and pressure system

In pressure systems the mode of flow that establishes itself in the pipeline may vary greatly, depending on a number of factors: gas velocity, settling rate of the material particles, loading, properties of the material, its frictional behaviour, alignment of the conveying pipeline (horizontal, vertical, presence of bends). Different types of installation are used, depending on the nature of the material to be handled, the conveying distance and the purpose of the conveying operation.

The loading (solids-to-gas ratio) μ denotes the ratio between the weight of solid matter to the weight of air and constitutes a criterion with which pneumatic conveying can be subdivided into three main ranges:

(1) low-density conveying for $\mu < 30$;
(2) high-density conveying for $\mu > 30$;
(3) fluidized or plug flow conveying with maximum attainable loadings.

With low-density conveying, the individual material particles are freely suspended in the conveying air, with few particle-to-particle collisions. This form of

pneumatic conveying has long been applied and is also most readily amenable to theoretical and experimental treatment. Granular materials are easier to handle in this range of flow than finely pulverized ones.

It has been found in practice that pneumatic conveying is more economical in proportion as the loading is higher. Although the mode of flow which characterizes high-density conveying is difficult to analyse theoretically, most pneumatic conveyors operate in this range. It extends from steady-state flow of particles fully suspended in the air stream through various intermediate modes, including unsteady ones, to steady-state fluidized flow and plug flow. With the latter, loadings of 250 and upwards are sometimes attainable. These various modes are indicated schematically in Fig. 22. With higher loading the conveying velocity decreases, while they conveying pressure increases. Finely pulverized materials which can readily be fluidized are more particularly suited for fluidized conveying, in which case the mixture of air and solids behaves in the manner of a fluid. On the other hand, granular materials which are difficult to fluidize are more suitable for plug flow in the sense of the material being pushed along as a quasi-solid "plug" of closely packed particles which do not move in relation to one another.

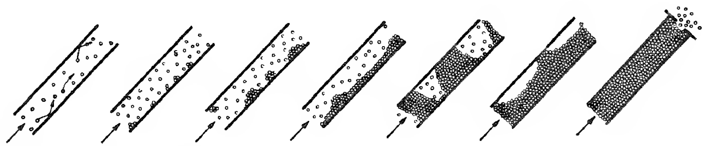


Fig. 22· Forms of flow

Another, and somewhat arbitrary, subdivision distinguishes low-pressure, medium-pressure and high-pressure conveying. Various pneumatic handling systems are reviewed in Table 20. In the column "conveying speed" the settling rate of material particles is denoted by w_f.

The feed devices for introducing the material into the conveying air stream, and the separators for removing it from the air stream, are important parts of the equipment. Feed arrangements are simplest in suction or vacuum systems, as illustrated in Fig. 23. Some of the conveying air flows through the bulk material into the suction nozzle, while part of the air is drawn directly into the nozzle through adjustable apertures by means which the loading (solids-to-air) ratio can be varied. With pressure systems the feed-in of the material presents more problems. For relatively low pressures (above atmospheric) in the pipeline it is possible to use injectors as feeding devices, operating on the principle of the jet pump (Fig. 24).

With these devices it is not possible to attain substantial throughput rates, as only the kinetic energy of the air jet is available for overcoming the back-pressure in the conveying pipeline. However, despite low handling capacity, this type of equipment has its uses for the removal of collected dust from filters and of cement spillage from sack packing machines, i.e., applications involving relatively small quantities and short conveying distances.

Table 20: Survey of pneumatic handling systems

designation			loading (–)	pressure p (mm w.g.)	max. conveying distance (m)	conveying speed v (m/s)	energy demand (kWh/t)	handling rate (t/h)	nature of material (mm)	remarks
low-density conveying	low-pressure conveying	suction systems	0–10	0–2000	up to 300	vertical 1.2–1.5 w_f, horizontal 2–2.5 w_f, ≈20–32 for fine-grained material up to 7 × w_f, and more	0.8–6	normal 3 to 50 max. 500	coarse and fine 0.5–50	steady-state conveying in suspension
		combined suction and pressure systems	10–30	2000 to 5000	300 to 600					
	medium-pressure conveying	pressure systems	10–30 20–50						0.5–50 5–200 µ	
high-density conveying		pressure systems	> 30	> 5000 to 3 bar	to 2000				coarse or fine	unsteady-state conveying possible formation of streamers, balling and plugging
	high-pressure conveying	fluidized conveying	up to 250	3 to 7 bar	up to 1500	5–20	≦ 0.8	up to 150	fine 1–20 µ	steady-state conveying; intermittent (batchwise) conveying only for short distances (avoud bends!)
		plug flow conveying	up to 1000 and more	high pressure	20	0.5–5	high	up to 50	coarse also moist	

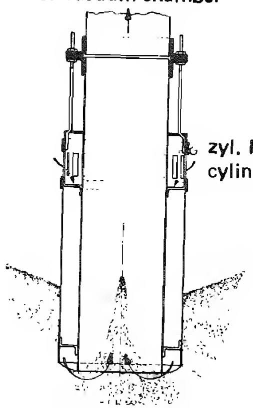


Fig. 23: Suctions intake nozzle

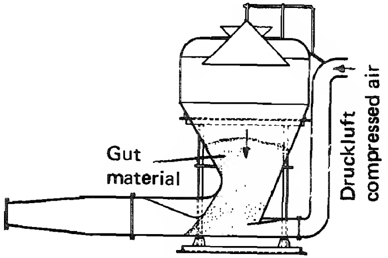


Fig. 24: Feed-in of material into a low pressure pneumatic conveying system

For higher pressures (up to 1.4 bar) so-called rotary gate (or rotary valve) feeders may be employed (Fig. 25), which function as air-locks. Similar devices may be used also for the discharge of the material from the pneumatic handling system. A special form of such feeders is intended for dealing with sticky materials which tend to remain adhering in the compartments or pockets of the rotor and is equipped with air jets that dislodge the material by blasts of air.
For even higher pressures, feed screws may be used for introducing the material into the conveying air (Fig. 26). Against their advantage of providing a continuous

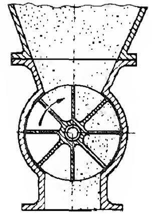


Fig. 25: Principle of a rotary gate feeder, enclosed type

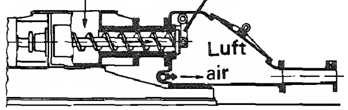


Fig. 26: Fuller pump for feed-in of materials into a high-pressure

feed must be set their high power requirement and heavy wear. The feed screw generally runs at high speed (750 – 1500 r.p.m.), the air sealing action being achieved by the screw and the material itself. The non-return valve at the end of the screw seals off the outlet when the screw is not feeding and is useful more particularly when blowing out the pipeline with compressed air.
An example of a screw-fed pneumatic conveying system is the Fuller pump, which can attain throughputs of about 200 m^3/hour and conveying distances normally ranging from about 40 to 200 m, though there are instances of as much as 1000 m. The loading is generally between 30 and 40%, but may be 80% in special cases. The compressed air is as a rule supplied by rotary compressors delivering air at between 1.0 and 2.5 atm (gauge pressure). Power consumption is in the range of 0.8 to 1.5 kWh/tonne per 100 m and is thus in excess of the requirements of mechanical handling systems of equivalent capacity.
The type of material handled by the screw feeder evidently plays a major part with regard to rates of wear. In finely powdered materials such as cement raw meal the amount of wear can be reduced by using higher screw speeds, this being made possible by the better sealing action of such materials. However, since wear cannot be obviated completely, the screw feeder should be so designed that all wearing parts are easily accessible and can be replaced by new ones in the shortest possible time.

The pitch of the screw is generally constant along the entire length. Sometimes, however, for dealing with readily compressible materials and widely varying input rates, a compressing screw is used, i.e., with decreasing pitch in the direction of conveying.
Where fine-grained and powdered materials have to be conveyed in a substantially vertical direction the feed-in arrangements may alternatively take the form of a vertically mounted vessel into which the material is introduced by a pneumatic trough conveyor. At the same time, air is blown into the vessel from underneath and serves to keep the material in a fluidized condition. In addition, conveying air is introduced through a central nozzle at the bottom. The column of material itself forms the air seal. An advantage of this vertical handling system in comparison with a bucket elevator is the lower initial cost, the absence of moving mechanical parts and the practically unlimited conveying height attainable. Also, it is adaptable to varying throughput rates and can be built to high capacity ratings (several hundred t/hour). The drawback is that for, say, 80 m conveying height the power consumption of about 0.8 kWh/t is something like 60% higher than that of an equivalent bucket elevator. Even so, there may be economic advantages in this method pneumatic handling if capital and maintenance costs of the installation are duly taken into account. Conveyors of this type are available from many major engineering manufacturers, e.g., the "Peters Airlift" (Fig. 27).
The pressure vessel pneumatic conveying system must also be mentioned. A special vessel (blow tank) is partly filled with the material to be conveyed, the feed inlet is closed, and compressed air is then introduced into the vessel, causing the mixture of material and air to be discharged through a bottom outlet into the conveying pipeline. High loadings (up to 200) and long conveying distances are attainable by this method. Power consumption is 0.5 – 1.0 kWh/t per 100 m. The operation is batchwise, i.e., intermittent, with intervals for refilling the vessels in parallel, one being filled while the other is discharging its contents into the pipeline. With fairly coarse-grained materials, simple settling hoppers or receivers are used for separating them from the air stream at the discharge point on the conveying pipeline. For finer materials it is preferable to use cyclone separators. Cyclones may moreover be installed as secondary collectors or dust arresters downstream of primary separators. The functioning principle of the cyclone is very simple: the air-and-solids mixture enters it tangentially and follows a spiral path in which the particles are flung outward (by centrifugal force) against the wall of the cyclone and fall by gravity to the collecting hopper at the bottom (Fig. 28).
Pneumatic trough conveyors (fluidizing conveyors, "airslides") are used for conveying finely divided materials over relatively short distances in slightly downward inclined paths, e.g., from a bin to a feeder. The operating principle of fluidization consists in aerating the material, as a result of which each particle becomes enveloped in a film of air which acts as a lubricant, as it were, so that the particles become almost frictionless in relation to one another and the material as a whole temporarily acquires "fluid" characteristics. More particularly, it can flow like water down an inclined surface.
The trough or duct of a fluidizing conveyor is generally of rectangular cross-section and is separated into an upper and a lower compartment by a longitudinal partition

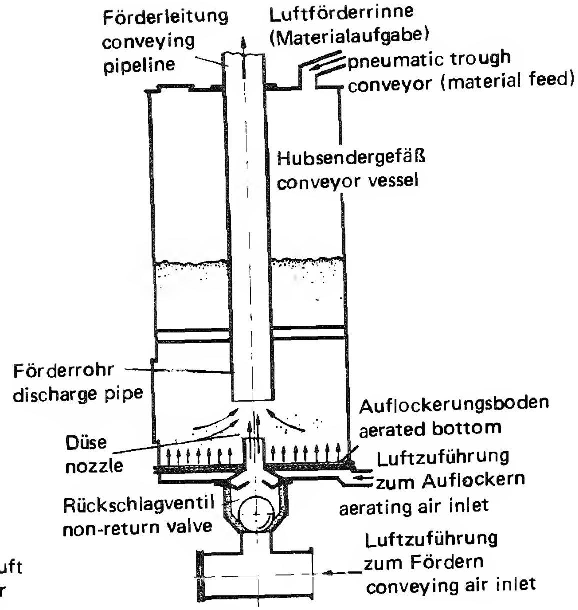


Fig. 27: Feed-in into vertical pneumatic conveyor

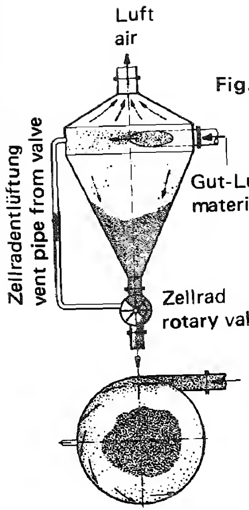


Fig. 28: Discharge cyclone for separating the conveyed material from the conveying medium

permeable to air. Compressed air is introduced into the lower compartment, which occupies about one-third of the overall cross-section of the trough, and flows through the partition into the upper compartment where it fluidizes the material. The latter should contain a sufficient proportion of fine particles in order to develop an adequate fluidizing action. The trough and its longitudinal partition have to be so designed that a substantially uniform air pressure is maintained along the entire length of the lower compartment. Very moderate air pressures, supplied by an ordinary fan, suffice for operating a pneumatic trough conveyor.

The handling rates attainable by these conveyors will depend on the width and the angle of inclination of the trough. As a rule, the angle is between 4 and 6 degrees, with a possible maximum of 10 degrees. With troughs ranging up to 1000 mm in width it is possible to attain throughputs of up to 2000 m^3/hour. These conveyors can, if necessary, be laid to curved alignments. Special slide gates and diverter switches can also be installed. With such devices the material can be transferred from one pneumatic trough conveyor to another or be discharged sideways directly into a bin or hopper (Fig. 29).

The bellows-type mobile loading spout for discharging fine-grained materials into bulk container vehicles (Fig. 30) is fed by a pneumatic trough conveyor. The spout can be raised and lowered and comprises a double bellows arrangement through which the material is passed in the inner bellows tube, while air is extracted

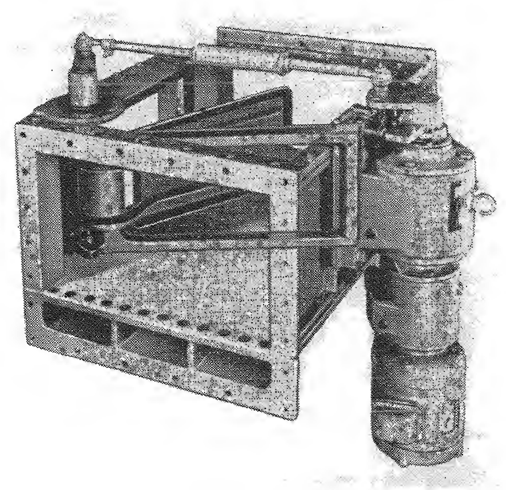


Fig. 29: Side discharge device or diverter switch for pneumatic trough conveyor

Table 21: Pneumatic trough conveyors

internal width	mm	100	200	315	400	500
construction depth	mm	100–200	180–250	200–330	300–400	350–410
raw meal handling rate	t/h	15	40	60–80	80–150	120–200
cement handling rate	t/h	20	45–60	70–100	95–180	145–250
air requirement/10 lin. m lfd. m	m^3/min	1.5–4.0	3.0–6.0	4.5–7.0	6.0–8.0	7.5–10.0
air pressure at fan	mm WS	250–500	250–500	250–400	250–400	250–400
power consumption per 10 lin m at full rate	kW	0.30 –0.40	0.35 –0.80	0.4 –0.90	0.63–1.28	0.7 –1.60

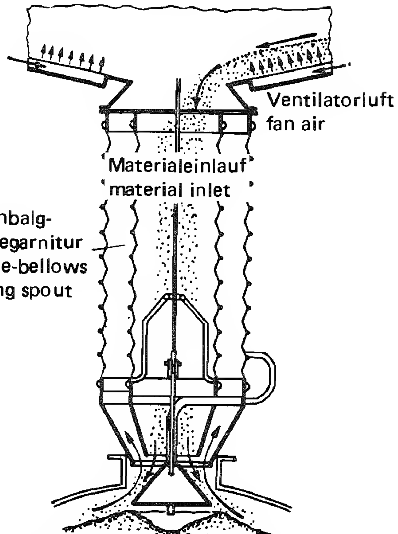


Fig. 30: Bellows-type mobile loading spout with dust control

through the annular space between the inner and outer bellows, so that dust nuisance to the environment is obviated. The tapered nozzle of the spout fits into the filling inlet of the vehicle and is provided with a valve for dust-free cut-off of the flow on completion of the loading operation. The whole spout assembly is mounted in an overhead carriage which can travel longitudinally a distance of 15 m, so that it can be moved to successive filling inlets on a long vehicle without having to move the vehicle itself. Loading rates of up to 250 t/hour are attainable.

VIII. Feeders

Cement manufacture is a continuous process requiring the regulated and proportioned supply of raw materials and uninterrupted discharge and removal of the products. Storage vessels such as hoppers, bins, bunkers and silos are provided

with various types of closing and discharge device – slide gates, pivoted gates, rotary gates, etc. – for controlling the outflow of the materials stored in them. With special arrangements the rate of discharge can be quantitatively controlled to dispense sufficiently precise quantities as and when required. These devices are especially appropriate for loading bulk materials into vehicles, suspension rail skips, etc. On the other hand, if continuous discharge at a controlled rate, e. g., for feeding a continuous conveyor, is required, the bin or silo outlet is equipped with a power-driven dispensing (flow regulating) device, so as to ensure that the conveyor will not be overfilled and thus function less efficiently or even suffer damage. Similarly, suitably regulated flow rates are essential for feeding material to screening systems, as the screens will not perform efficiently if they are overloaded.

Feeders for bulk materials in industry mostly have to function under rough service conditions. The materials themselves present a wide range in terms of particle size, moisture content and temperature (from $-35°C$ to $+700°C$). More particularly, they may consist of fine or coarse particles, possess good flow properties or be dust-like with unpredictable flow behaviour or be of sticky consistency, etc. For dealing with such a wide range of materials it is of course necessary to have a corresponding variety of feeding and discharging devices, duly suited to the properties of the material that each type has to cope with.

Practical experience has shown that a substantial proportion of faults arising with continuous conveying systems must be attrubuted to shortcomings in the design of the feeders used in conjunction with them. These devices and their correct installation should therefore always be given due attention. Table 22 can facilitate the correct choice of such equipment for dealing with specific materials. Since most irregularities in the rate of flow of the material to be handled arise from "bridging" or "arching" in discharging and feeding, a classification into ten grades of "awkwardness", i.e., difficulty in terms of flow and handling behaviour, has been proposed for bulk materials by Taubmann, the rating "10" being given to the class of materials with the most difficult properties.

Although most feeding devices are specifically designed for their function, many of them operate in accordance with the general principles applicable to continuous conveyors – chain, belt, screw and vibratory conveyors in their various forms – since feeders are, as a rule, specialized conveyors in their own right, except that they are often of more robust construction to resist the forces exerted by the column of material in the feed hopper from which they extract the material. Belt conveyors for this purpose are generally confined to fairly light materials fed at comparatively low rates. They are, however, frequently employed in the form of belt weigh feeders which are of variable-speed design to give controlled flow rates as required or which otherwise are designed to switch off automatically after a certain (weighed) quantity of material has been dispensed. Feeders of this type usually operate in combination with a vibratory conveyor into which the material is primarily discharged from the hopper or bin.

Continuous-flow conveyors, screw conveyors or apron conveyors are much more extensively used as handling devices for the extraction materials from bins, etc. The various forms of construction of these devices have already been discussed in the

Table 22: Bulk materials and appropriate feeders

	angle of repose	apron feeder	belt feeder	continuous-flow feeder	screw feeder	vibratory feeder	rotary gate feeder	chain feeder	table feeder	difficulty rating
ash (dry)	40–60°		×	×	×	×	×			2–3
ash (wet)	45–60°		×						×	5–7
barite	30–45°	×	×	×		×				2–5
dolomite	30–45°	×	×			×		×	×	2–5
iron sulphate	30–45°		×			×				6
ores (heavy)	40–50°	×				×		×	×	4
ores (light)	40–50°	×		×		×				4–5
fly-ash (dry)	20°			×	×		×			7–8
gypsum	30–45°		×	×	×					2–5
blastfurnace slag	30–45°		×			×			×	1,5–3
lime (slaked)	35°		×	×	×	×				6
hydrated lime	5°/60°						×			8–10
limestone	30–45°	×	×			×		×	×	2–4
gravel	30–45°		×			×			×	1,5–2,5
coal (normal)	35–45°	×	×	×	×	×			×	3
coal (very abrasive)	35–45°	×	×			×			×	4
coal slurry	–	×	×	×	×	×				5–6
pulverized coal	30–60°			×			×			6
coke	30–45°	×	×			×				4–5
magnesite	30–45°	×	×			×		×	×	3
raw meal (cement)	30–45°			×	×		×			6–8
soot	15–30°			×	×		×			7–9
salt (lumps)	30–45°	×	×			×			×	2–4
salt (common)	30–45°		×			×	×			3
sand (dry)	30–45°		×			×			×	2
sand (moist)	45–60°		×			×			×	3–6
fireclay	30–45°		×		×	×	×		×	2–5
talcum (crushed)	30–45°	×	×			×		×	×	2–4
talcum (powder)	30–45°		×	×	×	×	×			4–6
clay (loam)	45–60°	×	×			×			×	7,5–10
cement	5°/60°			×	×		×			6–8
cement clinker	30–45°	×	×			×			×	2–3
zinc white	30–60°				×		×			6–8
hot materials	30–45°	×				×			×	3–5,5
hygroscopic materials			×						×	5–9
sticky materials				×	×					6–8,5
granular materials			×	×	×	×		×		1–3
dust					×		×			5–8,5

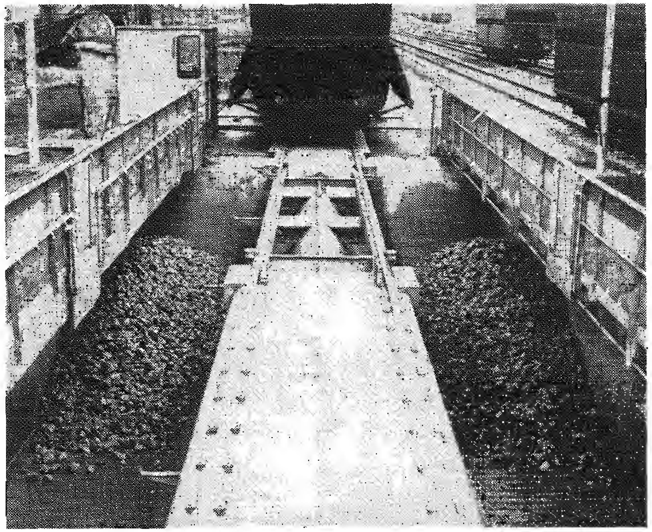
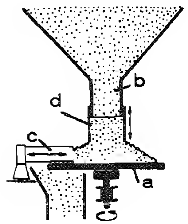


Fig. 31: Buckle-plate conveyors as loading hopper bottoms

preceding sections of this chapter. As a rule, the extracting conveyor is installed under a discharge hopper (Fig. 31) and directly withdraws part of the column of material resting on the conveyor. In the case of flight conveyors (scraper conveyors) and apron conveyors it is possible to vary the rate of discharge by changing the speed and/or the height of the bed of material deposited on the conveyor. If the discharge of material has to be temporarily stopped for maintenance or repairs to the extraction device, the column of material can be arrested by means of rods inserted through holes (on one or both sides of the outlet opening) so as to form a temporary grid.
A different principle is embodied in the rotary gate (or rotary vane) feeder, also known as the star feeder, which can perform an extracting and dispensing function and can moreover be sealed to serve as an air-lock gate in conjunction with pneumatic handling systems (Fig. 25). The flow rate through the feeder can be controlled by varying the speed of rotation These devices are of various designs and forms of construction suited to the flow behaviour and other properties of material to be handled. With non-sticky materials there is generally no problem, but if the materials are sticky or tend to cake, the rotary gate must be designed with special care, particularly as regards the shape of its rotating compartments. Thus, these may be semi-circular and of a depth suited to the material behaviour. Relatively thin-walled rotors, of welded construction and containing steel balls

have proved especially suitable for difficult materials. The tumbling motion of the balls keeps the rotor in a state of vibration which assists the material to fall out of the compartments.
A well-known type of feeder which has the advantages of good feed rate control, versatility and high operational reliability is the table feeder (or disc feeder), as shown in Fig. 32. It consists essentially of a rotating disc which is mounted under the bin or hopper outlet. The material flowing from the outlet onto the disc forms a heap whose size can be varied by raising or lowering an outer collar or sleeve on the outlet spout. The disc carries along material from underneath the heap and, by the action of a scraper on the disc, discharges it into a chute. Besides flow control by means of the collar and by means of the scraper (which can be extended farther into the heap or retracted, as required) there is a third possibility by varying the rotational speed of the disc. These feeding devices are of robust construction, which makes them heavy, so that their power consumption is relatively high.



a rotary table;
b hopper outlet;
c scraper;
d collar (can be raised and lowered)

Fig. 32: Table feeder

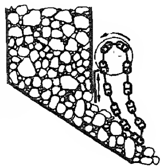


Fig. 33: Chain feeder

Bulk materials with very coarse and abrasive particles have to be discharged and fed by devices of a different kind. For example, Fig. 33 shows a chain feeder, which consists of a number of round-link chains mounted loosely side by side, suspended curtain-wise from a rotating drum which can move them. The chains should be so heavy that they prevent any material from flowing out of the hopper when they are at rest. The chain curtain should moreover be of sufficient width so that some of the chains also rest against the sides of the bed of material and thus arrest its flow. To discharge the material, the drum is rotated, allowing a certain quantity to slip through under the chains.

If coarse bulk material has to be preclassified, it is possible to use a combination of chain feeder and travelling grate functioning as a screen (Fig. 34). The finer particles that fall through the grate are collected in a hopper and discharged through a separate chute. The grate bars are self-cleaning on passing round the end pulleys. Preclassification in conjunction with material discharge can alternatively be achieved with a roller-bar grate (Fig. 35). The rollers are provided with cam-like projections which produce a heaving motion in the overlying material, which is thus prevented from choking the grate and moreover undergoes a certain amount of blending. Devices of this type are especially desirable for feeding heavily contaminated materials (or with excessive proportions of undersized particles) to crushing plants.

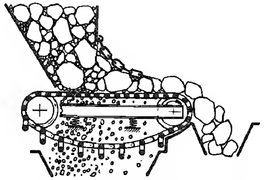


Fig. 34: Preclassifying feeder: combination of chain feeder and travelling grate

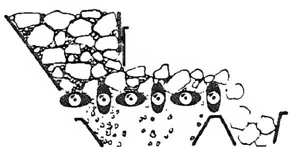


Fig. 35: Travelling grate

The discharge openings of long bunkers can most suitably take the shape of a continuous slot. A longitudinally mobile discharge carriage (Fig. 36) equipped with a horizontally rotating set of scraper blades removes the material from a shelf under the outlet slot and deposits it onto a belt conveyor mounted under the edge of the shelf. Devices of this general type are suitable for almost any bulk material which will remain lying at its natural angle of repose on the shelf, i.e., not be so "fluid" as to spill continuously out of the slot, but they are more particularly suitable for fairly sticky materials which are difficult to discharge by other methods. The drawback is that such installations are expensive in construction and operation.
Like continuous-flow conveyors, screw conveyors can likewise be installed directly under a bunker or bin and may be provided with several inlet and outlet openings. Since screw conveyors develop a kind of cutting and shredding action within the material, they are suitable also for difficult materials which tend to choke

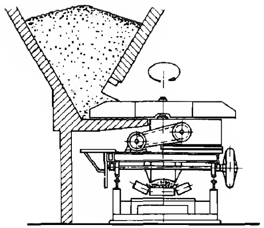


Fig. 36: Slot bunker with discharge carriage

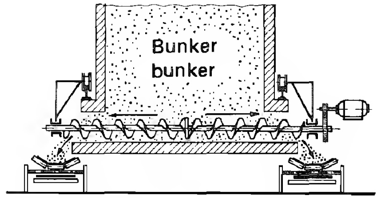


Fig. 37: Mobile discharging screw

other types of conveyor by matting or felting. In such cases the bunker is provided with virtually vertical side walls, while the extractor screws extend fully across the bottom discharge opening (Fig. 37).
Feeding methods for swing bucket elevators include devices for dispensing predetermined quantities of material to the individual buckets. Feeders for pneumatic conveyors have already been described in the relevant sections of this chapter.
Finally, mention must be made of a discharge system which simultaneously blends the materials extracted from several bunkers, bins, etc. disposed one behind another. An apron conveyor installed under the row of outlets extracts a layer of material from each, the desired proportions of the mix components being regulated by varying the depth of the layer discharged from each outlet. Mixing or blending is done by a paddle shaft at the discharge point of the apron conveyor. This system is more particularly suitable for the blending of various grades of coal. If very intensive mixing is required, a twin-shaft mixer can be installed in addition to the paddle shaft (Fig. 38).

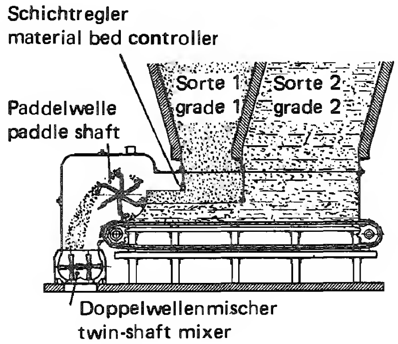


Fig. 38: Coal mixing installation

Alternatively, paddle shafts can be installed at the head end of a discharge conveyor for sticky materials such as loam or chalk. With this arrangement the material is, as it were, chopped or sliced off as it emerges from the hopper outlet and is then fed in a uniform layer to the conveyor (Fig. 39).

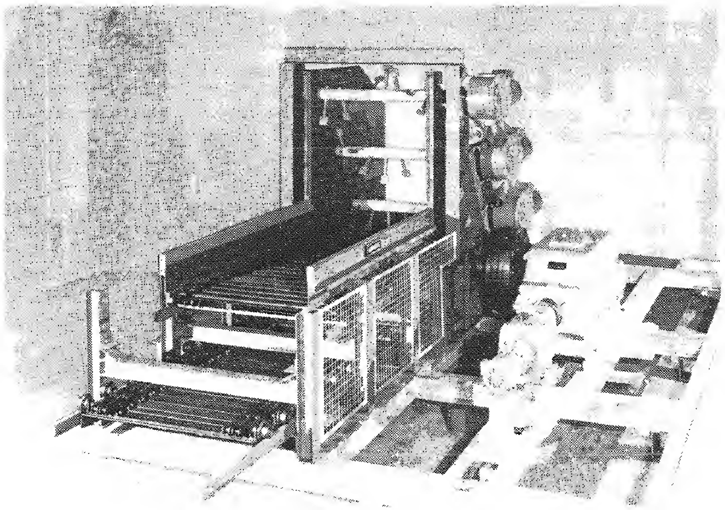


Fig. 39: Paddle shafts for the discharge of sticky material from a bunker

IX. Weighing equipment

From the description of the various feeding devices in the preceding section it appears that many of these are capable of extracting material at a uniform rate from a bin, bunker or hopper and thus provide the basis for at least a volumetrically controlled and measured flow. In automated industrial processes, however, more exacting requirements as to the accuracy of flow measurement are often applied – beyond the capability of those devices. More particularly, precise weighing of quantities is required. For many years the only practicable method of doing this was to transfer the materials from continuous conveyors into automatically functioning weigh hoppers. This not only involved additional handling, but these

manipulations were also liable have an adverse effect on friable materials. Besides, the weigh hoppers were bulky pieces of equipment and therefore often difficult to accommodate in the conveying path.

The problem was solved by the development of suitable belt conveyor type continuous weighing devices (belt weighers) which can be mounted in the supporting frame of a normal belt conveyor and form an integral part thereof The actual weighing unit is connected by a lever system to a special roller set which functions as a "weigh-bridge" for the material passing over it on the belt. The lever is directly attached to a temperature-compensated load cell which measures the weight continuously and is protected from damage by overloading. The weighing unit can be separately calibrated under static or dynamic conditions. The load on the measuring roller is transmitted to a weigh-beam which actuates a totalizing device. The weighing operation involves no vertical travel of the roller, i.e., the latter remains at a constant height. The totalizer is driven by the bottom strand (return run) of the belt via a friction roller, so that correct measurement is obtained even with varying belt speeds. The system can give a direct indication of the conveying rate (in t/hours) at any given time as well as recording the total quantity conveyed on the belt in a certain length of time. Belt weighers of this type attain ± 1 per cent accuracy of measurement.

Whereas the belt weigher measures the rate of flow as it happens to be, it may in other cases be necessary to supply the material at an accurately weight-controlled rate, i. e., a specified weight per unit of time. This is done by the weigh belt feeder, also known by such names as belt weigh feeder or conveyor belt scale. (The designation "proportioning belt feeder" is, strictly speaking, appropriate only if the machine, possibly in combination with others of the same type, dispenses one of the components of a mixture at a specified rate).

The feeder consists basically of a belt weigher with electric speed control. The value of such machines for producing mixtures correctly proportioned by weight in various industrial processes is obvious (Fig. 40).

Weigh belt feeders can be installed as extractor belts under the outlets of bins, hoppers, etc. Such a belt may be driven at a constant speed. In order to achieve a constant rate of discharge (in terms of weight per unit of time) the arithmetical product of the belt speed (v) and belt loading (q) must be kept constant. The simplest arrangement consists in utilizing the weight-measuring roller under the belt as the control element for adjusting the device which regulates the depth of material discharged onto the belt at the bin outlet. Unfortunately, this method is often too inaccurate to meet the feed control requirements in some processes, because the material depth adjusting devices are too insensitive or, if the material contains oversize particles and small depths are required, tend to become partly choked, so that irregular functioning occurs.

This snag can be overcome by giving the outlet opening the largest possible cross-sectional dimensions and varying the speed of the belt (Fig. 41). With this method the accuracy of feeding depends only on the accuracy with which the weight-measuring roller operates. This in turn, however, depends on the flexibility of the belt and on the tensile force acting in it. A drawback is that the resistance encountered by the material on discharge cannot be controlled, while the changes

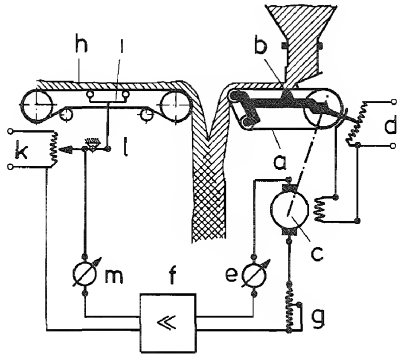


Fig. 40: Combination of a belt weigher with a weigh belt feeder

a swivelling weigh and extraction belt of the feeder; b pivot; c variable-speed drive motor; d potentiometer pick-off for bulk density compensation (y control); e weigh belt feeder power indication; f amplifier; g potentiometer for adjusting the weight ratio; h belt conveyor; i belt weigher; k potentiometer pick-off for fine adjustment of weigh belt feeder; l transmission lever; m belt conveyor power indication

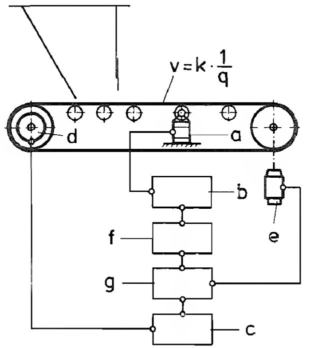


Fig. 41: Weigh belt feeder, single-section type

a weighing unit; b measuring bridge; c controller; d variable-speed motor; e tacho-generator; f intermediate storage for measured values; g product computing device

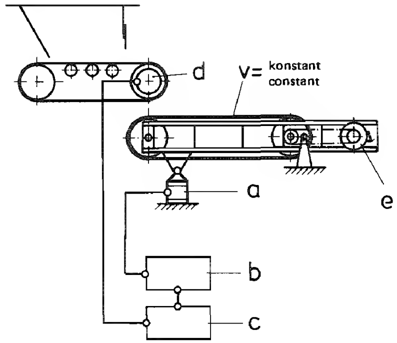


Fig. 42: Weigh belt feeder, two-section type

a weighing unit; b measuring bridge; c controller; d variable-speed motor; e synchronous motor

in belt flexibility due to ageing of the belt also constitute an uncertainty. To compensate for these problems, rather sophisticated and expensive measuring and control equipment has to be used, as appears from Fig. 41.

On the other hand, the system illustrated in Fig. 42 is much simpler, involving merely a weighing unit and a controlled feeder completely separate from it. Any of the familiar types of bin or hopper discharge devices can be used in conjunction with this equipment.

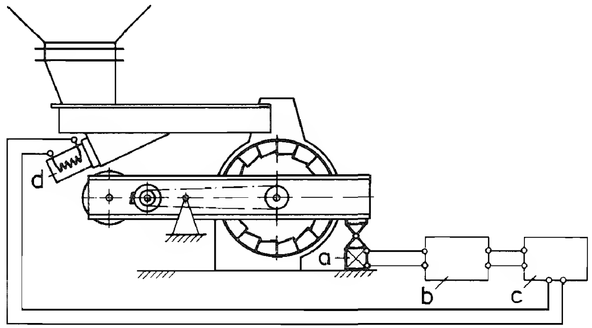


Fig. 43: Feeding system with rotor-type measuring device and controlled vibratory conveyor

a weighing unit; b measuring bridge; c controller; d variable-speed motor

Continuous weighing equipment incorporating a rubber belt mounted on measuring rollers is not suitable for dealing with materials with irregular flow behaviour and a tendency to "flushing". Nor can it be used for very hot materials. For the latter, similar weighers, but incorporating steel apron conveyors instead of belts, have been successfully introduced. Alternatively, a so-called rotor weighing unit has been developed for such materials (Fig. 43). Instead of a belt conveyor there is a rotor somewhat resembling a water-wheel with compartments into which the material to be weighed is admitted. The flow rate is controlled by the speed of the rotor driven by a synchronous electric motor. The rotor is mounted on the weighing unit, so that in this respect its operating principle is no different from that of the belt weigher with its measuring roller. The material from the bin or hopper is fed to the weigher through any suitable type of gate or other discharge device.

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G. Process engineering and automation

By G. Schmiedgen

I. General

Modern cement works are controlled and monitored from control stations which may be decentralized or centralized. The trend is towards, the central control room or control centre, from where one or more production lines can be operated with the minimum of personnel.

With this technology the control room personnel usually have no direct contact with the process to be monitored. They have to rely on suitable means of communication with the individual machines and other units of process equipment. Such means are collectively categorized as process engineering and automation. In this context the term "process control system" is used.

What does this term signify?

It comprises the following functions:

- measuring
- interlocking
- monitoring
- operating
- closed loop controlling
- computing.

The electrical manufacturing industry supplies suitable equipment and systems for performing these functions. The necessary engineering may be undertaken by the

customers themselves, by independent consultants, by the suppliers of the process machinery or by the suppliers of the process control system.
As a result of the impetuous developments in semiconductor technology, there are now efficient and inexpensive electronic components and modules available with which it is becoming increasingly possible to shift the duties of process control and monitoring from human attendants to the process control system. At the same time, however, there are limits of economy and effectiveness to be considered in pursuing this trend.
A process control system must primarily ensure that the production process is safeguarded and maintained. In addition, it must reliably provide the means of operator/process communication and make possible the connection of higher-order system components for optimization duties.
Communication of the human operator with the process he has to control is very important. To provide automatic responses to each and every demand made by the process would be economically impracticable and also undesirable from the point of view of reliability of the system. Human intervention remains necessary, subject to the condition that it does not impose too heavy a strain on the operating personnel involved. The control system must meaningfully select, prepare and present the requisite information from the process, so that even in critical phases of operation no wrong interventions due to overburdening of human judgment and response capacity will occur.
Reliability of the process control system and efficiency of its design and construction are of major influence upon operational dependability and economy in cement manufacture. It should be borne in mind that we are here concerned with a chain of functions or units of equipment. This chain begins at the sensor or limit switch. It comprises adaptation, transmission, monitoring and processing of the signals from the industrial process, and it ends at the valve, variable drive or other final control element. It also includes the machines and other associated units of equipment. This chain is only as reliable as its weakest link. The designer of modern process control systems for the cement industry has to take due account of these matters in selecting the equipment and determining the duties it has to perform.
These general considerations concerning a complete process control system also apply to the individual functional groups for measuring, controlling or monitoring.
The information given in this chapter lays no claim to be an exhaustive account – which would be outside the scope of this book anyway. Its object is, rather, to provide the cement specialist with an outline the principles and review the present "state of the art" in this field of technology.

II. Measurement and process control

This section is concerned only with the actual process variables. Purely electrical quantities (current, voltage, power, cos φ) are not dealt with, as these do not directly impinge on the cement engineer's sphere of duties.

1 Measurement

By "measurement" in the present context is understood the quantitative detection of characteristic technological values, their conversion into analog electric signals, their transmission and their evaluation. In cement manufacture the problems of measured data collecting are centred mainly on choosing the suitable measuring points and most reliable measuring methods. With the large rotating units of plant involved, it is mostly possible only to perform indirect measurements of the internal processes and then only at a limited number of points. For example, the physical and chemical reactions taking place inside a rotary kiln can be monitored and assessed only in terms of auxiliary variables (temperature, pressure, gas analysis) which are accessible for measurement at the inlet and the outlet end. Under these conditions, electrical engineering and instrument technology must co-operate closely with the practical experience of the "men on the job" in order to assess just how representative and reproducible certain changes in a measured variable are, so as to justify technological inferences drawn from them. Complicating factors are the large quantities of dust swept along in the gases, the high temperatures, the abrasive action and the chemical aggressiveness of the material being processed. In view of these circumstances, it is inevitable that practically no measuring point – i.e., a point where a sensor or detector for monitoring the process is installed – will continue to function impeccably for any great length of time.
On the contrary, measuring points require regular maintenance, for only in this way can they go on giving reliable information on the technological relationships over reasonably long periods. Accordingly, in trying to judge the merits of a measuring system, it is necessary to take account of its convenience of maintenance, the service life of the sensor, its adjustability (manual or automatic) and the possibility of supervising its functioning. Programmable systems (computers, motor control and process control systems), which will be discussed more fully further on in this chapter, offer considerable scope for the automatic monitoring (and calibration, if necessary) of measuring points while the plant is in operation, but such arrangements cannot enable the services of instrument maintenance personnel in the cement works to be dispensed with.
The analog signals picked up by the sensor are usually very low-powered (millivolts, microamps) and therefore highly sensitive to disturbing influences. In practice, a suitable technique consists in converting these weak signals into impressed direct-current signals in the immediate vicinity of the sensor, so that the transmission of the measured variables (telemetering) can be effected through cables, free from problems arising from disturbing influences. The two types of signal in common present-day use are the so-called "dead zero" (0–20 mA) or "live zero" (4–20 mA) The dead zero signal facilitates computing operations in analog technology, which are important especially in multi-loop control systems without a process computer. With the live zero signal it is possible to employ two-wire cables to the transducer and easier monitoring of the transducer and signal transmission circuit. A measured value below 4 mA indicates a fault. In principle, it makes no difference which of these two alternative signal types is adopted, but it is important to ensure that consistent signals are employed throughout a cement works. This eases the problem of spares inventory simplifies the work of the

maintenance gang and enables different system components to be interconnected without complications. A recent trend has been to convert the direct-current signal in the control room (after transmission from the plant) into a voltage signal (e.g., 0–10 V), which makes it easier to distribute the measured variable among several analysing or indicating instruments.

Instruments for the detection of **temperature** are thermocouples, resistance thermometers and radiation pyrometers. With the first two types of instrument the sensor comes into direct contact with the medium whose temperature is to be measured, so that this part of the instrument is liable to suffer wear or damage. On the other hand, the radiation pyrometer picks up thermal radiation emitted from the surface of a body, i.e., the instrument does not come into direct contact with the medium, but it must nevertheless be protected from excessive heat, which might harm its electronic equipment, or from dust deposits forming on its lens. It is obviously important that the pyrometer should really receive the radiation that is representative of the temperature to be measured. The optimum direction of sighting the instrument must be adjusted at the outset.

Depending on the range of measurement, the purpose for which the results are required, etc., there are various methods of measuring pressure, whether as **absolute pressure, gauge pressure** (above atmospheric), **vacuum pressure** or **differential pressure.** With correct choice and suitable mounting of the instruments, they will operate reliably. Measurements in heavily dust-laden atmospheres are liable to cause problems with choking of the tubes leading to the instruments. In designing and installing the latter it is therefore necessary to ensure that there are proper access openings or devices for the regular cleaning of the pressure measuring equipment. Automatically operating devices for cleaning by compressed air jets have generally proved effective.

For repairs or replacement of faulty pressure transducers, adequate shut-off devices (valves, etc.) must be provided, which should of course be readily accessible to maintenance personnel.

Differential pressure measurement is employed also for the detection of possible blockage of gas flow passages, for monitoring the loading of tube mills, and (in conjunction with various types of equipment) for measuring the **flow of gases.** Gas flow measurement is of major importance, being required for stabilizing the conditions in the rotary kiln, the grate cooler or the air separator. In certain cases it may necessitate extra constructional cost. With the large dimensions of gas ducts in present-day use, the classic pressure differential rate-of-flow measurement with a venturi tube or flow nozzle involves building very substantial structures. Besides, adequate lengths of duct in which steady flow conditions can develop will have to be provided upstream and downstream of the venturi or nozzle. Attempts to measure dust-laden gas flow by means of static pressure tubes have hitherto proved unsuccessful. On the other hand, interesting possibilities are offered by new methods of calculating the rate of flow with the aid of a computer using auxiliary measured variables and the characteristic curve of the fan. But here, too, the dust content of the gas, and changes in the fan characteristic due to wear or build-up of deposits, can cause problems.

For measuring the **flow of solids** (particulate bulk materials) various types of

continuous measuring equipment or belt weighers are used, depending on the required accuracy. Weigh belt feeders serve also as flow regulating devices, dispensing the material to a process at a specified rate or proportioning the components of a mixture. For these applications the equipment must comprise, in addition to the actual weighing system, a finely adjustable dispensing device suited to the flow properties of the materials.

For the **flow of liquids** there are various kinds of metering equipment, while inductive methods can also successfully be used for suitable liquids (conductivity not less than 0.5 μS/cm).

Statutory requirements for the prevention of environmental pollution are making it increasingly necessary to install instruments for monitoring the emission of pollutants. **Smoke density** measuring instruments are based on the absorption of light by the smoke. A light source directs a beam of light across the flue onto a photoelectric cell which generates a measuring voltage, the value of which depends on the degree of light absorption and thus provides a measure of the dust concentration. In modern systems such monitoring devices are themselves monitored by checking units which integrate the pollutant emission levels over successive periods.

Rapid and reliable **analysis of the kiln exit gases** is very important. The measuring points of particular interest are:

- CO content of the exit gases before the electrostatic precipitator: for this check for protecting the precipitator the important thing is to obtain the measured result quickly.
- CO_2 content before and after the preheater: these measurements give information on the degree of decarbonation (calcination) of the raw meal being fed to the kiln.
- O_2 content at the kiln inlet: this analysis enables the air excess in the combustion process to be kept within acceptable limits from the point of view of firing efficiency.

Unfortunately, none of these measurements are free from problems, so that they can only tentatively – and with a sufficiently wide margin of uncertainty with regard to the theoretically optimum value – be utilized for process monitoring and control. The problems begin with the extraction of gas for analysis. The object is to obtain a properly representative sample from the gas flow in the kiln, which in large plants will be in the region of 800000 m^3/hour.

Besides, the sampling system should be so arranged that the effect of inleaking air upon the results of the measurements is kept down to a minimum. Problems in this respect are liable to occur especially in the feed end housing that forms the transition from the preheater to the kiln.

As a rule, the instruments are accommodated in a steel cabinet which also contains the gas cooler, condensate trap, filter, diaphragm pump, etc. and is installed as close to the sampling point as possible. In this way, controlled preparatory processing of the gas for analysis is achieved and, what is even more important, short gas flow paths, thus helping to obtain rapid analysis results. These aspects must be given particular attention at the initial design stage.

X-ray analysis is especially significant in connection with cement manufacture,

because it enables rapid analysis of the raw materials and of the finished product to be carried out under plant operating conditions, thus providing the basis for reliable quality control. In the main, X-ray fluorescence analysis (X-ray spectrometry) is used, a method in which the sample to be analysed is irradiated with X-rays and is thereby made to emit its own characteristic radiation, the wavelengths of which give information on the nature of the chemical elements in the sample, while the intensity of the radiation gives an indication of the proportions in which these elements are present.

There are several types of X-ray spectrometer equipment·

- multichannel spectrometer in which several (in cement works usually from five to eight) elements are measured simultaneously,
- sequential spectrometer with a radiation detection channel which can be moved through an angular range to certain measuring positions, thus enabling different elements to be measured successively;
- continuous spectrometer, through which the material for analysis flows in a stream which is continuously analysed; this instrumentation can be used only in a fixed location and for a specific purpose, whereas the other two types of X-ray spectrometer are universal instruments.

In connection with cement manufacture, X-ray diffraction analysis (X-ray diffractometry) is used for the determination of free lime in cement clinker.

Correct preparation of the samples for analysis is important, as this is of major influence of the accuracy of the results.

Various techniques are used for **monitoring** and **measuring the levels** of materials in bins, hoppers and silos. For responding to predetermined fixed limits (full, half full, empty) devices such as diaphragm switches, probes of various kinds. or sensors for radiation from radioactive emitters are employed. Most of these devices suffer to a greater or lesser extent from problems due to wear and/or to caking or clogging with moist or sticky materials, while the use of radioisotopes requires due compliance with the relevant safety regulations.

Suspended probes which are lowered automatically from the roof of a silo or bin and which respond by automatically switching off on coming into contact with the material are suitable devices for measuring the level at any given time. The actual contents of the silo can be estimated from the distance travelled by the probe, making due allowance for the angle of repose of the material.

Alternatively, with modern techniques, the entire contents of a bin can be weighed by means of load cells. These devices should be installed as permanent features in the construction of new bins.

Because of the comparatively elaborate equipment required in relation to the technological benefits achieved, the automated monitoring and **measurement of moisture** content has hitherto been applied only on a limited scale in the cement industry. Mostly, estimates of moisture content are based on the measurement of auxiliary variables, e.g., temperature of exit gas.

Automatic **fineness measuring** instruments have hitherto also found only limited application. The determination of fineness, optimum granulometric composition (particle size distribution) and bulk density (of clinker) by suitable automatic means offers scope for future development of instrument technology.

2 Closed loop control

The purpose of process control is to adjust the actual value of a process variable so as to make it, and keep it, equal to a given desired value (or set point). In modern industrial processes the automatic closed-loop control (or feedback control) principle is extensively applied: any deviation from the desired value is fed back into the control system and acts in such a way as to reduce the deviation of the controlled quantity from the standard value.

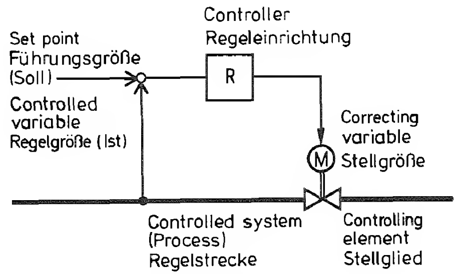

Fig. 1: Closed loop control principle

Basically, a control loop can be conceived as comprising a measuring element (detecting element, sensor) which measures the actual value of the controlled variable and transmits a signal (for example, in the form of an electric current) to a controller. The latter can be defined as a device which holds the controlled system (the process) at a desired level or state by comparing the actual value with the desired value; it transmits an actuating signal which causes a final control element (or correcting element) to take corrective action to restore the controlled variable to the desired value.

In principle, the controller functions like a human process operator who watches the dial of an instrument indicating the actual value of a process variable, e.g., a rate of flow. When the pointer moves away from the desired value, the operator corrects this by adjusting a valve (the correcting element) so as to restore the flow rate to its desired value. In modern industry the control functions are automated, using electronic equipment. Pneumatic control systems, which were used to some extent in the cement industry in the past, are now virtually obsolete.

Controllers with switching-action and with continuous output signals

The distinction is based on the nature of the output signal from the controller. In the European cement industry the "switching-action" principle is mostly applied. For motor-actuated correcting elements (e.g., control valves) it offers the least

expensive solution because it acts directly upon the power contactors of the servomotors or actuators. The controller giving a continuous output signal will require the interposition of a storage unit and additional position controller.

Compact controllers and controller modules

Another distinction is based on the form of construction of the controller: the "compact" controller as opposed to the controller of modular design, i.e., composed of individual "building blocks" (modules).
The "compact" type of controller can be installed directly in a control desk or switchboard (Fig. 2). Besides elements for performing the actual controller function it comprises the associated monitoring and operating equipment such as the set-point adjuster, manual override, and indicators for the controlled variable, the correcting element setting and the control deviation (the deviation of the actual value from the desired value). The advantage is the compactness of the unit, but this is attended by the disadvantage of limited adaptability of the circuitry. Complex multi-loop control systems, as well as direct control interventions (e.g., in connection with starting and stopping sequences), cannot suitably be achieved. In such cases the modular controller is to be preferred. The individual modules (Fig. 3), forming self-contained assemblies, are combined with one another according to the control problem that has to be solved and are mounted and wired

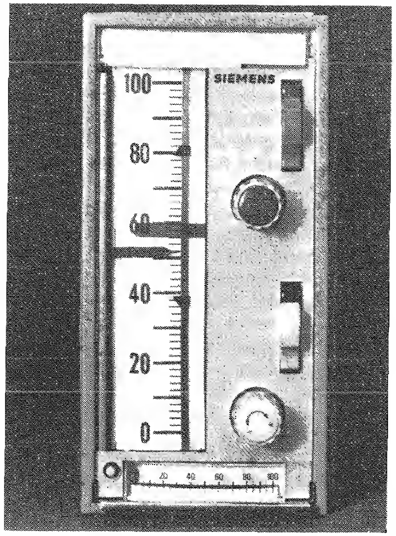

Fig. 2:
Compact controller (Siemens)

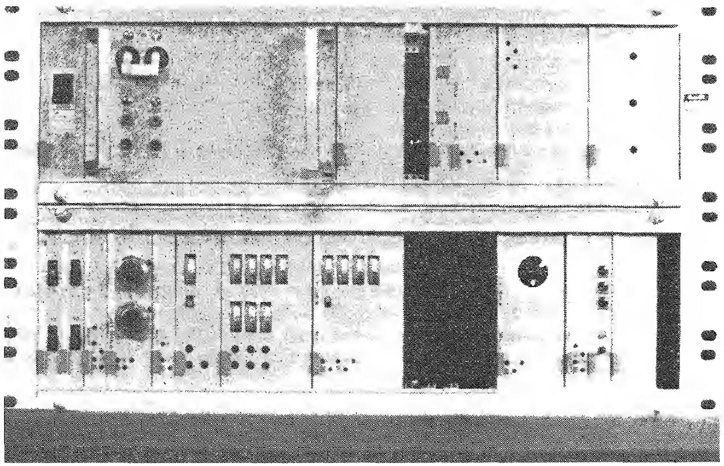

Fig. 3: Controller module (Siemens)

together in a mounting frame. Automatic closed-loop control, computing, motor (sequence and interlock) control and monitoring functions can be combined. The controller equipment can be installed at any desired point, not necessarily in the immediate vicinity of the control desk or indeed of the control room. Only the actual operating switches, keys, pushbuttons, etc. and indicating instruments are mounted on the desk.
These conventional controllers are subject to limitations in terms of circuit technology. A wider range of possibilities is offered by programmable control systems. With the introduction of a process computer it becomes possible to employ DDC control (direct digital control). In that case the controller function (or the control algorithm) is no longer performed by electric circuitry and switching actions, but is stored as a mathematical program in the computer. With these arrangements it is possible to carry out elaborate calculations for the control deviation or for the adjustment of the manipulated variable and also, for complicated controlled systems, to program the computer to perform the functions of two or more controllers which are brought into operation to suit the varying requirements of the process at any given time.
The following types of control are to be distinguished:

- **Pure DDC control:** In this method, control is done entirely by computer. If the computer develops a fault, the control system fails. It is an economical method which is especially suitable for slow control adjustments (e.g., temperature control), because these can be stabilized by manual intervention

in the event of computer trouble. DDC systems are now successfully used for rotary kilns equipped with planetary coolers.

– **DDC back-up control** offers the same advantages as pure DDC, but a conventional controller connected in parallel and automatically kept in step with the DDC system can smoothly take over the duties of the latter in the event of computer trouble.
– **Set-point control** or supervision: The computer determines the set point (desired value) for a subordinated controller which is in continuous operation.

When the process computer was introduced in the mid-1960s, the above methods were – mostly for reasons of economy – applied in the form of digital multiple control: one computer served a number of controllers. However, developments in electronics technology have in recent years resulted in increasingly advanced miniaturization of the components, attended with very substantial reductions in cost. The microprocessor constitutes what is at present the final stage in this evolution, opening up new possibilities for the application of programmable systems.

It is now possible, at acceptable cost, to use several decentralized computers for control duties. Programmable control systems as well as discrete individual controllers embodying microprocessors are available for the purpose. The problems associated with this technology lie not so much in the comparatively inexpensive and versatile hardware it employs. The cost associated with designing, programming, incorporating and adapting the software to achieve a reliable system of control for economical cement manufacture is the major consideration. The significance of these often underrated problems will be considered somewhat more fully later on.

In cement manufacture the raw mix control and throughput optimization of the raw mills occupy a special position. Acceptable solutions to both these problems became available only with the advent of the process computer, as these control functions involve elaborate mathematical calculations. Besides, both these controlled systems are characterized – for technical reasons – by long running times. The principle of sampled-data control operated by the computer can advantageously be applied in these cases, i.e., the control system intervenes only cyclically in the process and, after output of new weigher settings, allows the grinding plant to adjust to the altered conditions before another correction is applied.

Automatic safeguarding of the quality of the raw material can be achieved in three stages:

– controlling the stockpiling of the material components in the blending bed;
– blending control at the raw mill;
– controlling and balancing the homogenizing plant.

For performing these duties, small representative samples (about 100 g each) have to be taken cyclically from the overall flow of crushed stone upstream of the blending bed and/or from the flow of raw meal downstream of the grinding plant, with rapid chemical analysis by X-ray spectrometer or neutron activation, in conjunction with remote-controlled weigh belt feeders. Depending on the

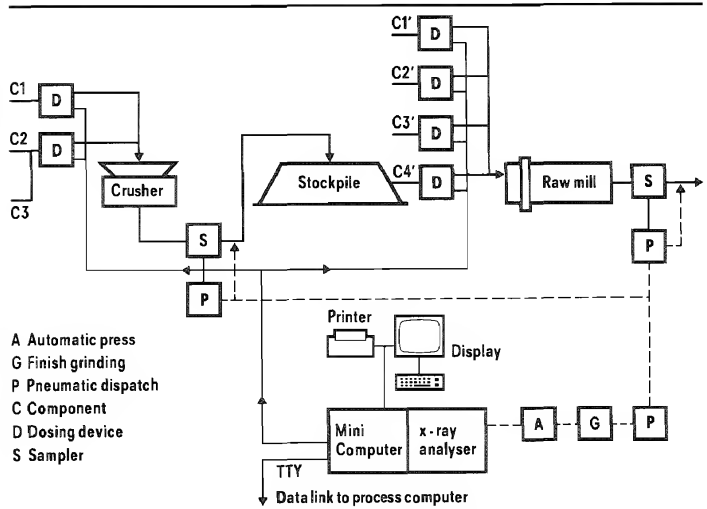

Fig. 4: Raw material mixing and blending control system (Siemens)

chemical composition of the raw materials, it will have to be decided in any given case whether and to what extent this three-stage quality control is necessary and economically advantageous. Automatic blending control at the raw mill, however, is now part of the established state of the art and will only exceptionally be dispensed with in new installations.

Various mathematical procedures are available for optimization of the raw mill throughput. They are nearly always based on the cascade control principle, in which the subordinate controller stabilizes the feed rate, while the master controller strives to achieve optimization by analysing the signals it receives and seeking the most favourable operating point for the mill to cope with varying grindability of the raw material. The respective methods differ in the choice of the measured variables they employ, their processing and the mathematical algorithms applied.

The plant engineer should bear in mind that the control system comprises, besides the detecting element and the controller, also the final control element, which is generally a feature of the electrical and/or mechanical engineering equipment of the plant. It is this equipment that is of major importance in deciding the efficiency of a control method. Thus, if it lacks the necessary reserve capacity, accuracy or reliability, the value of the control system as a whole will be impaired.

III. Programmable controllers

In a cement works the sequence and interlock control of the numerous motors and associated machinery is of major importance. As distinct from automatic process control, these plant motor control functions are basically performed on the open-loop principle and play an important part in ensuring reliable and economic operation of the production lines.
In a purely quantitative sense, too, motor control technology is a major feature in modern cement manufacture. For establishing and performing the sequencing, positioning and interlocking functions something like five to six thousand binary signals (acknowledgement and status signals, warning and alarm signals, position signals) have to be handled and processed. Interventions in the motor control system involve 1000–1500 commands (binary outputs).
Electronic motor (sequence and interlock) control systems have come into widespread use in the cement industry in recent years. The real breakthrough came with the introduction of programmable controllers. The systems now employed are robust, reliable and easy to handle. In contrast with relay technology, there are no moving parts that suffer wear.
How does a programmable controller function and what are its advantages?
Such a controller is what is known a bit processor, functioning in principle in the same manner as a computer, but differing from it in some important respects.

- The information is processed only as individual bits (as distinct from the computer, which performs word or byte processing).*)
- The controller does not require an operating system.
- The number of instructions or commands that can be carried out is smaller.
- The fewer instructions are, however, more effectively geared to the special control duty to be performed.
- The hardware of the controller is simpler.
- The programming of the controller is simple and easy to learn.

The basic features of a programmable controller are represented in Fig. 5.
The program store contains the control program, i.e., the programmed control sequence and interlock conditions. The control unit cyclically reads the contents of the individual cells of the program store and processes the stored control instructions. In this way signals are monitored and linked, time steps are initiated and evaluated, and drive motors are switched on and off. The control unit operates very rapidly, taking only 2–4 microseconds for processing an instruction.
The time steps perform the same functions as do programmable time-lag relays in more conventional technology. The marker stores are used for the storage of information arising from the process activities. The signal input and output serve to connect the controller to the plant which is to be controlled.

*) "Bit" is an abbreviation of "binary digit", conceived as a unit of information equal to one binary decision involving the two digits (0 and 1) in binary notation. "Byte" is a set of binary digits conceived as a unit and forming a subdivision of a "word"

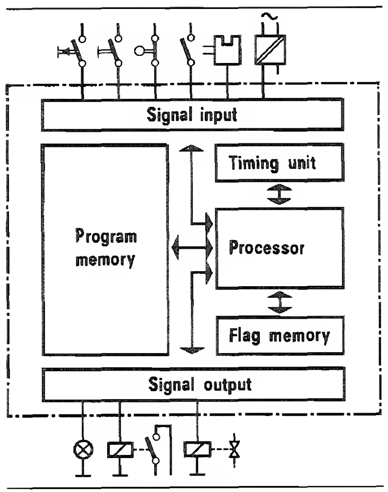

Fig. 5: Block diagram of programmable controller

The programs are "read in" with the aid of a programming apparatus and retained in the program store. Modifications or additions to the program can be effected in the same way. Fig. 6 shows a simple interlock control system based on the use of relays, on wired electronic units and on programmable control respectively. The three methods are identical as regards the functions they perform. In the program store of the programmable control each control instruction occupies one memory cell. Only the middle of the program list is contained in the store.
The figures in the left-hand part of the list denote the addresses of the individual cells; the letters on the right facilitate identification of the signals. The notation for the instructions is easy to understand; for example: UE 5 means "and input 5", OE 21 means "or input 21", SA 10 means "set output 10", etc. (these symbols are abbreviations of the German "Und, Eingang, Ausgang")
The flexibility of this programmable control system is illustrated in Fig. 7. Suppose that a change is to be introduced in that, for the signal D, an opening contact is to be applied instead of a closing contact. With relay control or with wired electronic control equipment it will, to carry out this modification, be necessary to alter the wired connections and possibly to install one or more additional components. On the other hand, with the programmable controller it is merely necessary to change the contents of memory cell 103. To do this, the instruction OEN 21 meaning (in German) "or input not 21" simply has to be typed in through the programming unit. This change can be effected in a matter of minutes, even while the control system is in operation.

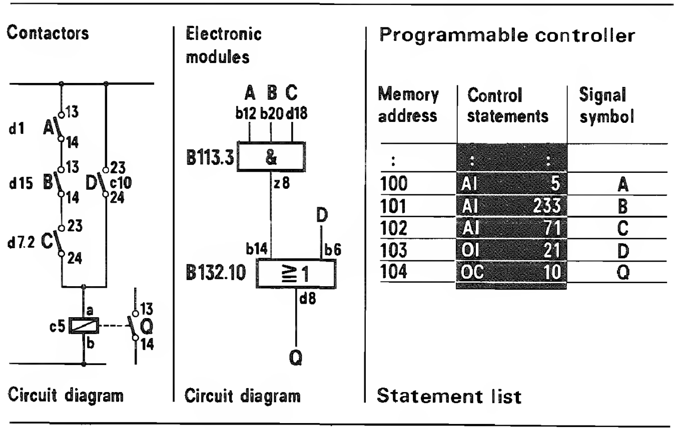

Fig. 6: Comparison between hard-wired control systems and programmed systems

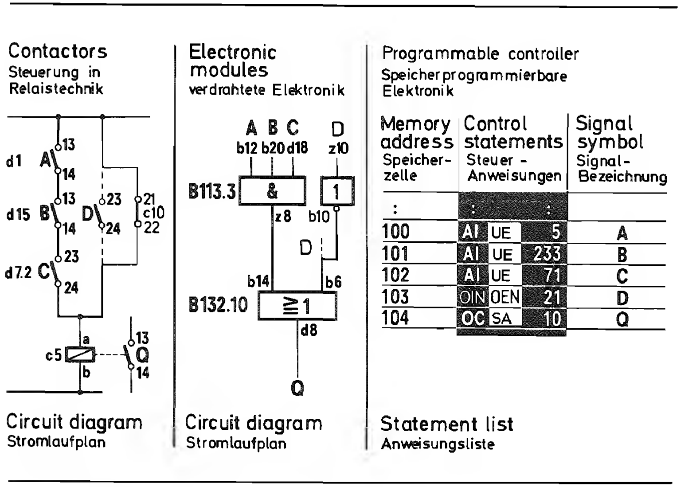

Fig. 7: Carrying out a change

For the read-in of the whole program or for carrying out more extensive changes, the programming units are equipped with cassette recorders.

The programming envisaged here is done with so-called mnemonic notation. However, controllers are available which alternatively enable the programming to be done in graphic form by means of a video screen in a functional diagram or ladder diagram representation. The processing of the program in the controller, as well as the advantageous flexibility in making changes, are based on the same principles as those applicable to mnemonic notation, however. With a programming unit directly associated with the controller it is thus possible to make changes conveniently and quickly, thus avoiding time-consuming detours via a computing centre for introducing program changes.

The advantages offered by programmable controllers can be summarized as follows:

- Substantially more complex control logic can be built up.
- The signals from the plant can easily be processed to provide the operating personnel with conveniently presented information.
- The whole control system can be built up with greater clarity of layout.
- Aids for fault locating and servicing, which are supplied along with efficient controllers, make it easier to track down the causes of trouble and thus shorten plant downtime.

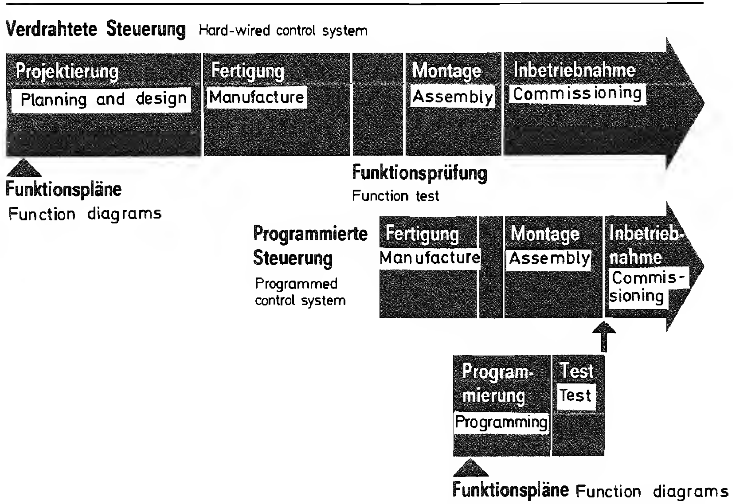

Fig. 8: Schedule charts for hard-wired control systems and programmed control systems

- The whole sequence of activities for introducing a control system, from the design stage to the commissioning of the equipment and process machinery, can be accomplished much faster and with fewer errors when programmable controllers are used. See Fig. 8.

This diagram shows the familiar procedure associated with setting up a control system based on hard-wired technology. Design, manufacture, functional testing (checkout), assembly and commissioning are operations which are carried out consecutively. The activities cannot begin until the functional diagrams are available. Errors affecting any stage in the sequence of activities will affect and delay the subsequent stages.
With programmable technology the procedure is very different. Hardware and software can be produced simultaneously. After the necessary inputs and outputs have been decided, the equipment can be mass-produced, tested, assembled and commissioned. At the same time the process engineering and technological design can proceed. Programming commences much later, so that the amount of errors and changes is greatly reduced. Moreover, the commissioning of the control system together with the plant is accomplished much more speedily, since adaptive adjustments are effected through the programming unit.

IV. Monitoring and operation

As already stated, it is not possible to do without the human operator and his decisions in any process control system. To enable him to intervene effectively, the man at the control desk requires reliable and readily intelligible information that will enable the situation to be quickly analysed at any given time. This information arrives in the form of binary or analog signals from the process. It has to be suitably processed and converted for presentation in a meaningful form and for possible automatic evaluation. This data processing is done at the control centre, where all the signals converge from the various parts of the plant. For the sake of reliable monitoring it is important to ensure that equivalent items of information receive uniform and consistent treatment.
The necessary indicating and operating equipment is assembled on or in desks, wall panels, cabinets or combinations of such units. Although the central control room is often the showpiece of the cement works, efficiency of layout should always be the prime consideration. Mostly the indicating and the operating elements are incorporated as integral features of a flow chart of the whole plant. In this way a visual interrelation between the information and its source within the plant is established, thus making for greater clarity of presentation. Besides inexpensive sheet-metal units for accommodating the equipment, there are other forms of construction embodying flexible grid layouts comprising mosaic elements of plastic (e.g., 25 mm × 25 mm) (Figs. 9, 10) or prefabricated sheet-metal elements (e.g., 24 mm × 48 mm). Grid technology is characterized in that the front of the wall mimic diagram or control panel is composed of individual elements which are equipped with operating pushbuttons and indicator lamps and can be provided with printed-on flow chart symbols. With their aid a very compact and convenient

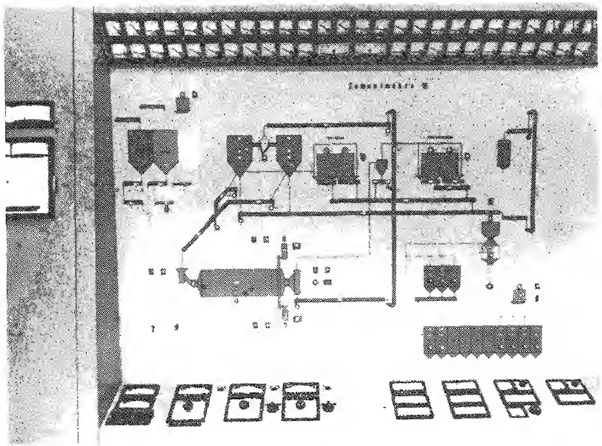

Fig. 9a: Control panel with mosaic mimic diagram

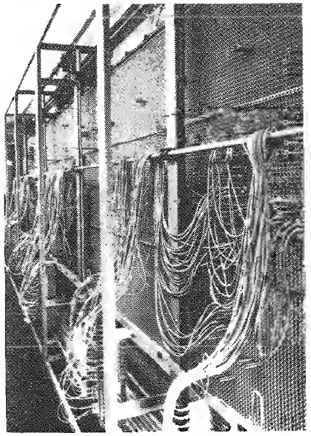

Fig. 9b: Mosaic mimic diagram (reverse side)

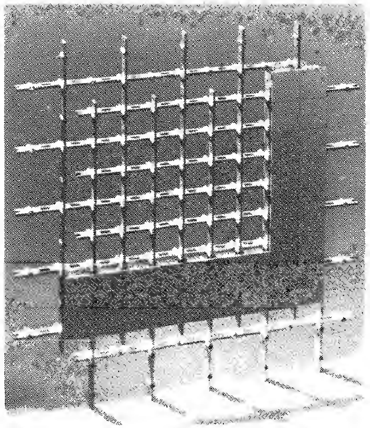

Fig. 9c: Mosaic mimic diagram (detail)

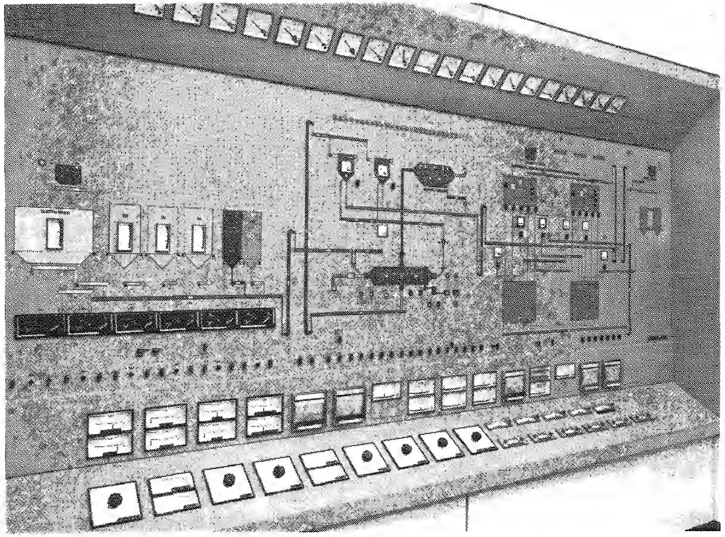

Fig. 10: Control panel with mosaic mimic diagram

layout of the control room can be obtained, as in Figs. 11 a and b. Any changes in the instrumentation or plant flow chart can easily be made by changing the appropriate grid elements without adversely affecting the overall arrangement. Measuring instruments and recording devices can likewise be conveniently accommodated within the flow chart.

Moving-coil measuring units are increasingly used for indicating and for recording instruments, the advantages being easier adjustment and simpler spares inventory.

The introduction of programmable systems has resulted in major changes in the arrangements used in the control room. Information is now presented through monitor display units consecutively, not simultaneously as in earlier systems. Further details on this technology are given in Section V.2 under "Computerized control centre".

An important condition for the reliable and economical supervision of a cement works and thus for its profitable operation is the operating and signalling logic design employed. It should be uniformly and consistently applied to all parts of the plant and be properly suited to the working methods of the control room staff and

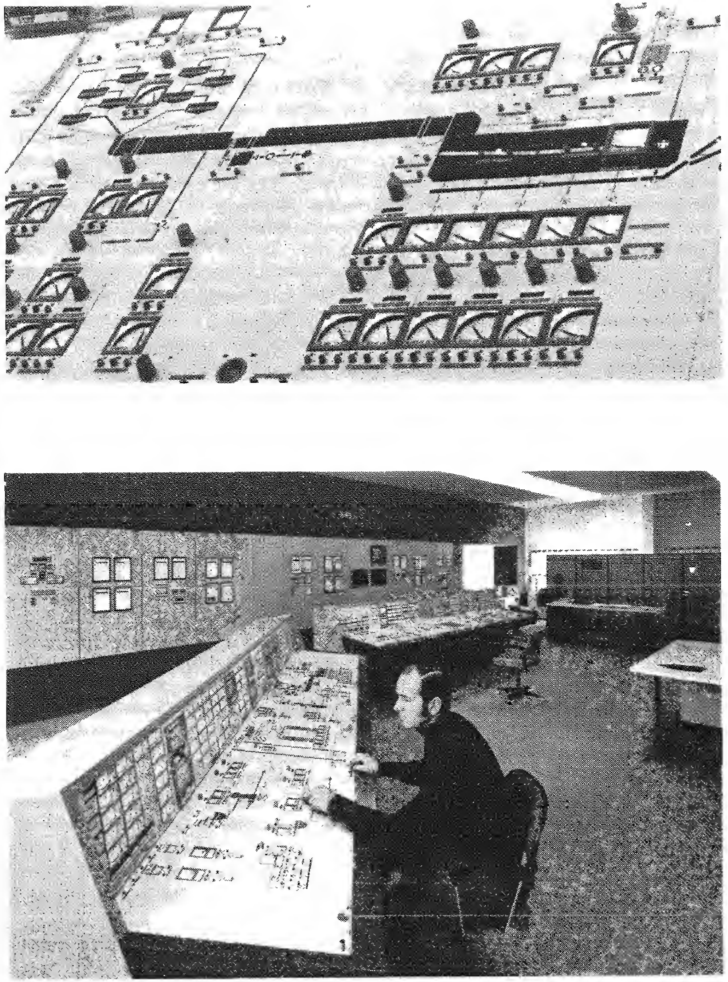

Fig. 11 a and b: Central control room with compact control desk (Siemens)

the repair and maintenance personnel. In planning the control system there is often a tendency to cut corners in this respect for the sake of a cheaper solution. This is liable to result in wrong decisions made in critical operating situations and in more time spent on fault-locating, repairs and restarting – with the attendant reduction in plant utilization time.

Anthropotechnical investigations concerning the stresses to which control room staff are subjected have shown that too much information offered simultaneously cannot be properly grasped and is liable to overburden the persons involved, with the risk that they will make wrong decisions. The most favourable solution is provided by a system which offers the process information in a form and manner conducive to correct decision-making.
During normal plant operation, the monitoring and operating functions can be limited to a few systematically selected items of information or interventions. Deviations which point to incipient disturbances or faults can then be conveniently identified side by side with the normal information data supplied.
Depending on how important any particular fault is, the operator in the control room may then automatically be given detailed information or he may call up such information if he requires it. Guided by this detailed information, the operator will then make the appropriate decision to cope with the situation. The information he is given corresponds to the various operating conditions, such as starting and stopping the plant, normal running, maintenance, fault locating and repairs. Good design of the signalling system will, in modern installations, enable the operator to judge whether a mechanic or an electrician is needed to remedy the fault brought to his attention.
A well designed operating and signalling system is backed up by a well conceived system for the identification of the various machines and other units of the plant, as well as the measuring points and signals, for the cement works as a whole. Guidance on these matters is to be found, inter alia, in the relevant Standards (DIN 40719, Sheets 2 and 40; DIN 19227, Sheet 1; ISO Publication 113-2).
Plant documentation is also important for economically running a cement works. It should likewise be uniform and consistent for the plant as a whole, should conform to the operating and signalling logic applied, comprise the actual state of the plant at any given time, and give readily available assistance to personnel for monitoring, maintenance and repair. It is now possible – and has already been applied in cement works – to use a computer for the documentation of the whole process control system. With this technique it is possible to produce from the same input data a number of different documents appropriate to various working phases associated with the plant. For example, the documents for erection are prepared differently from those for fault locating. The updating of documents after changes have been made is also facilitated with this method.

For the sake of completeness it should be mentioned that a modern control centre also comprises telecommunication equipment, such as:

- television system for the observation of critical sections of the plant, e.g., the burning zone and the kiln inlet;
- telephone system;

- intercommunication system (intercom) to enable the control room to keep in touch with the personnel in the plant itself.

Besides these various communication facilities the control centre comprises equipment for automatic back-up duties and for data conversion.
Normally, analog measured values are – depending on their importance – indicated on individual instruments or selectively (by means of a selector switch) on a shared instrument.
Some of the measured variables must additionally be monitored to ensure that they remain within permitted limits (e.g., temperatures of machinery bearings, CO content of exit gas). If the limits are exceeded, signals will have to be initiated and automatic intervention in the process effected. The signals for these purposes can be generated by measuring instruments or recording instruments equipped with adjustable limit contacts. More suitable, however, are separate limit monitoring devices which enable several upper and lower limit values to be freely selected and set. To each analog value that has to be monitored is assigned a printed circuit module which serves also as a transducer and sends out amplified signals (20 mA or 10 V) for further utilization at the control centre. By means of test and setting modules the respective limit values can be set individually without the associated measuring element having to be in actual operation.
The limit monitoring messages and fault messages from the plant are fed to an electronic processing system which receives and stores the information and provides acoustic and visual signalling (alarm, indicating lamps, etc.) to bring incoming messages of faults or other off-normal conditions to the operator's attention. Also, it is possible subsequently to distinguish between the first signal arising from the source of trouble and subsequent signals arising as a consequence thereof.
An electronic counting system may be provided as an extra feature, for the logging and summing of production values (e.g., quantities of clinker produced, bin levels, etc.) and of consumption values (e.g., oil or electric energy consumed).
If programmable systems are used, the functions of the limiting monitoring, message processing and/or counting systems can be performed by the programmable equipment.

V. Process computers

1 Development and use of process computers

In the cement industry the first process computers were installed in the 1960s and were nearly always assigned the task of raw mix control. In this way it was possible, under objectively monitored conditions, to ensure unvarying quality of the raw meal and thus to maintain definite input conditions for the subsequent production stages. As a result, many technological problems affecting the units of plant further along in the process were obviated or at least eased. Nowadays all process computers used in cement works have at least this mix control duty to perform.

After this successful start, it was in due course attempted to use computer technology also for solving other problems associated with cement manufacture. It turned out, however, that all efforts to solve these problems by means of mathematical models and making full use of the high efficiency of modern computers were doomed to failure, caused by the awkward practical conditions in the cement works. Only when it was realized that a stepwise adaptive procedure could lead to better solutions properly geared to practical reality was the breakthrough achieved which made possible the effective use of the computer for duties besides mix control. The modern approach more particularly takes advantage of the objectivity and flexibility of the process computer. Supported by a suitable software system, this method of solution enables plant operating experience to be optimally utilized and reproducible results to be obtained.

The various duties for which process computers have hitherto been used in cement manufacture are described below. This is simply an enumeration of solutions, without attempting any evaluation of priorities, which differ from one case to another. All these solutions have been actually applied in practice, but not – so far as the present author is aware – all in interrelated conjunction with one another in one and the same cement works. In general, success is achieved by meaningful restraint in making use of the possibilities and in stepwise realization of the computerized control schemes.

- Control of activities in the quarry with a view to calculating short, medium and long-term economic utilization of the available raw materials.
- Controlling the crusher for maximum throughput, taking due account of the operational requirements of low mechanical wear and minimum downtime.
- Controlling the stockpiling of materials in the blending bed, so as to ensure that the raw grinding plant always has suitable prehomogenized material in sufficient quantity at its disposal. The effectiveness of this control is to a great extent dependent on the choice of mechanical equipment for the blending bed and also on whether or not a crushed stone sampling system (an expensive feature) is installed.
- Raw mill control, in accordance with the following priorities:
  - chemical composition of the raw meal;
  - residual moisture content of the raw meal;
  - stabilization of the mill loading (filling ratio);
  - optimization of the throughput.
- Controlling the homogenizing and/or raw meal silos and keeping quantity records (embodied in balance reports) for these installations. These duties must, in control engineering terms, be viewed in connection with the control of the chemical composition of the raw meal.
- Controlling the rotary kiln. This is an especially interesting field of duties, since the complex features of kiln running cannot be properly stabilized in all operating phases by means of conventional analog measuring and control equipment. Digital control methods based on the process computer offer advantageous possibilities, but their successful application in practice involves some expenditure on regular attention and maintenance of the relevant measuring elements and correcting elements.

- Quantity records and balance reports of the clinker store.
- Controlling the finish grinding mill:
  - cooling the mill product;
  - fineness;
  - input of mix proportions for interground constituents;
  - stabilization of the mill loading (filling ratio);
  - optimization of the throughput;
  - optimization of the use of additives.
- Monitoring and keeping quantity records (with balance reports) of the cement silos.
- Controlling the cement loading and despatch operations. This range of duties is gaining in importance, because smooth and trouble-free despatch procedure is very important in view of the ever increasing capacities of cement works. The process computer can more particularly perform the following functions:
  - accepting and storing the despatch orders;
  - directing the vehicles and the despatch operations;
  - supervising and controlling the loading (tare weight, gross weight, grade of cement, etc.);
  - issuing the delivery notes;
  - producing the invoices or direct debits;
  - preparing and transmitting the despatch data for a higher-ranking commercial data processor;
  - quantity records and balance reports;
  - statistics.

  For the purpose of back-up and in order to enhance operational reliability, some of these duties in connection with despatch are sometimes assigned to programmable controllers that in turn are linked to the computer, which undertakes those operations involving much computational work.
- Logging and collecting the operating data, such as production quantities, material and energy consumption, machine running times and downtimes, time spent on repair or maintenance work, and supervising the stores for spare parts and process materials. Supervision of arrival and departure of personnel ("clocking in/out") can also be computerized.
- Commercial duties, such as customer statistics, bookkeeping and wages accounting. Normally, separate computer systems are installed for these two last-mentioned duties, but it is quite possible – and has indeed already successfully been done – to assign them to the process computer. Alternatively this computer may be used merely as a data collecting and data preparation device, the data thus acquired being then transmitted to a commercial data processor for treatment.
- General process control and monitoring of plant operation. These duties comprise every section of a cement works and make for clear-cut and rational works management. More particularly, they are:
  - acquiring the measured variables;
  - monitoring the measured variables;

- calculating specific characteristics or index values;
- signalling normal plant functions;
- signalling off-normal conditions or faults,
- output of processed information;
- logging;
- storing values for retrospective reference;
- fault analysis.

By solving these problems and performing these duties the process computer can complement the conventional instrumentation in the control room and supply the plant operating and managing staff with objective information. Here, too, sensible restraint is important. If the process computer puts out too much information, the advantage will soon turn into a disadvantage. The aim should always be to obtain as much clear insight as possible into the process events with the least amount of information compatible with achieving that aim. Comprehensive anthropotechnical investigations and the developments achieved in semiconductor technology, which have made better and better equipment available at lower cost, have resulted in the "computerized control centre".

2 Computerized control centre

With this technology the dialogue between the human operator in the control room and the process events in his charge are supported and guided by the process computer (Fig. 12). The necessary safety functions are automatically performed by subordinated systems. The man in charge still makes the operational decisions, but the computer so prepares the requisite information as to present him with clear-cut conditions on which he can decide and avoid overburdening him with too much information. In this way he will not be put under too heavy a strain, so that he is unlikely to initiate wrong interventions even in critical phases of plant operation.
In the control room this technology entails the change-over from "parallel" presentation of information (all the items of information displayed or indicated simultaneously side by side on an array of instruments and indicator lights mounted on long panels) to "consecutive" presentation in which the necessary values or messages are displayed sequentially on video units.
The **graphical display unit** supersedes the flow chart. The whole cement works is split up into a number of images, each comprising a technological section of the plant. These images can be called up as and when required. They comprise superimposed analog measured values which are continuously updated and which take the place of conventional indicating instruments in the control room (Fig. 13). These video images can be composed of up to seven colours. Off-normal conditions in any part of the process can be clearly displayed by changes of colour and/or flashing. The images can be built up on a function-related basis, e.g., an image of any particular plant section may contain different items of information for the plant in normal operation and for the starting or stopping phase (Fig. 14). The image structure and range of symbols to be used can, for major systems, be selected to suit the customer's specific requirements.

Fig. 12: Computerized control centre with operator's desk (Siemens)

The **curve display unit** (Fig. 15) supersedes the conventional recording instrument. All the analog measured variables connected to the process computer and the calculated characteristic values can be represented (to various scales) on this display unit. Modern units of this kind can simultaneously show up to seven curves (graphs) in different colours. Besides, several curves can be selected and assembled into a combined display. Records of past events can likewise be shown on the video screen for "post mortem" verification. The equipment supplied by some manufacturers moreover enables the flow chart sections to be shown with superimposed curves on one and the same display unit. While this is certainly an interesting technique, care must be taken that the clarity of presentation and informative value of the images are not diminished.
The functions of the indicator panel in conventional technology are performed by the **alphanumeric display unit.** The plant operating messages and fault indications are represented in clear text with time read-out and flashing light. Acoustic signal and message acknowledgment procedures are similar to those in conventional systems. It is advantageous to employ two display units for a plant

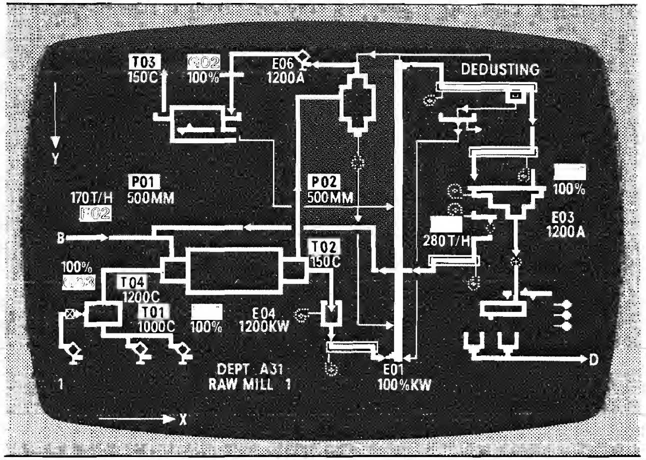

Fig. 13: Graphical display unit showing flow diagram with superimposed analog values

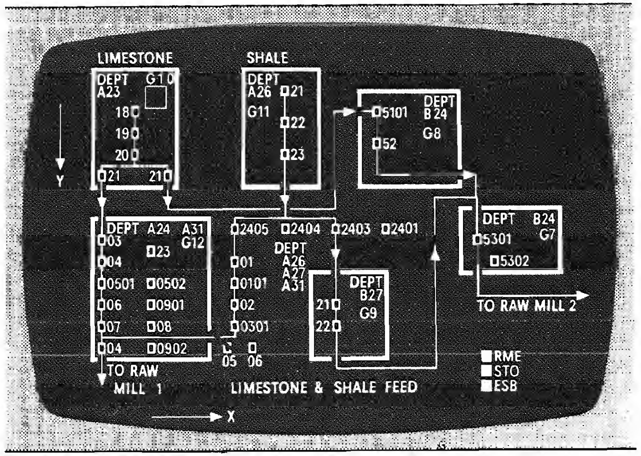

Fig. 14: Graphical display unit showing sequencing scheme

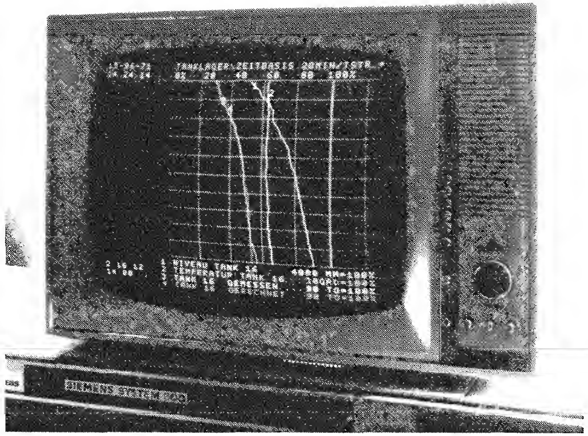

Fig. 15: Display unit showing curves

one for incoming messages as and when they arise, the other for the retrospective examination of earlier information awaiting assessment. These various messages and indications are sorted according to the respective sections of plant, so that, for example, the operator in the control room can check all this information in a conveniently presented form before starting a raw grinding plant. Any unremedied faults are thus detected before the start is initiated and can be put right.

Information displayed on the video screen can, if desired, be reproduced in the form of a document by means of a **hard-copy device**. The items of information are stored, with date and time of day, enabling critical process situations and other details to be retrieved for future analysis.

Printers are output devices which convert data into printed form. They are used for the printout of shift, day, month, fault and plant operation records. Stock inventories, statistical analyses and similar results of mathematical calculations can also be produced in print by such devices.

The various functions of the control centre can advantageously be performed through **dialogue keyboards**. The various plant sections (e.g., the raw mill), functions (e.g., starting procedure) and units of equipment (e.g., graphical display unit) are each assigned certain keys or pushbuttons, so that the operator in the control room can – for performing a specific function – obtain the desired information by pressing these keys. This does not require any special knowledge of computer technology; the keyboard can be provided with inscriptions in any language or with appropriate symbols.

So-called **light pens** are photo-electric devices which can be used for direct intervention in the above-mentioned control centre functions or in the process activities themselves via the display unit. Their function is supplementary to that of the dialogue keyboard or they may entirely take the place of the latter. The pen can be used to activate the computer to change or modify the images displayed on the screen, in accordance with movements made by the operator. Errors of operation are prevented by suitable interlock precautions in the computer. In some systems the light pen can actually be used to build up images specifically suited to the user's requirements directly on the display unit.
For input of set points or characteristic values the dialogue keyboard is complemented by an alphanumeric keyboard. These inputs can be introduced either directly into the flow chart of a section of the plant or by means of a special form. Before these inputs take effect, the computer checks them with regard to plausibility and reliability, so that erroneous inputs are substantially eliminated.

3 Hardware and software

The functions of a process computer are performed by the co-operation of hardware and software.
The term hardware applies to the physical units making up the computer system, i. e., the apparatus. In choosing a computer it is generally not too difficult, with the guidance provided by technical specifications and manufacturers' tenders, to select the hardware suitable for the intended duties. However, under present conditions, the hardware accounts for less than 50% of the capital expenditure on a process computer system, and this cost proportion is likely to decrease still further in relation to the cost of the software. The latter comprises the technological engineering and, especially, the programs that can be used on the computer system in question.
For assessing the suitability of a process computer system and its economic effects it is essential to check the programs and the aids for planning, commissioning and adapting the software which are available from the manufacturer of the computer. Misjudgements in these matters are liable to prove expensive and adversely affect the efficiency of the process computer.

Software can be produced by the following methods.

- Programming in machine-oriented assembler language: In terms of storage space requirements and run time this method can result in effective programs. It does, however, require specialized knowledge of the hardware and of the programming language employed. Changes and adjustments require programming and compiler runs in a computer centre.
- Programming in problem-oriented compiler language: The programming languages used for the purpose (e.g., FORTRAN, ALGOL, BASIC COBOL) have been developed for handling problems of certain general types and are machine-independent, i.e., they are used to describe a program without regard to any particular machine coding system. Normally an existing compiler program (e.g., in FORTRAN), after suitable translation in the computer centre, should be capable of execution on any make of computer. In practice, there are restrictions, however. This programming requires practically no knowledge of the computer hardware, and learning the programming language is much easier than when an assembler language is used. On the other hand, the programs are less favourable than assembler programs as regards storage space and run time.
- Programming with user-oriented system programs: This programming technique, which allows of planning with technology related concepts, is a development of recent years. The computer manufacturer supplies a comprehensive software system consisting of well-tried software modules (e.g., PID controller, limit monitoring) and a system frame. The modules are deposited in the computer once and for all, and are not changed by the user. Producing the user program consists only in interconnecting these modules and in providing them with the necessary technological parameters. These actions do not involve any interventions in the program modules and therefore require no special programming knowledge or skill. It is not so much programming as planning, which is done in a language that the technologist can directly understand. The great advantage of this technique is that adjustments and changes in the user program can be effected on-line while the plant is in operation.

Characteristic of all programming methods is that, with increasing comfort of programming, planning, operation and servicing, the storage space requirements for the software increase. The associated increase in hardware cost is, however, more than compensated by the savings effected as a result of flexible and simple handling during commissioning and implementation.

4 Microprocessors

With so-called one-chip processors, modern semiconductor technology offers an inexpensive module which can perform all the functions of the central processor of a process computer. As a result, many new fields of application for computer-based solutions are being opened up, and the end of this development is not in sight. Hence the introduction of microprocessors into the control of cement manufacturing plant obviously suggests itself, and has indeed been successfully accomplished for certain special purposes. It should be borne in mind, however, that with the use of a process computer in a cement works rather less than 10% of the capital cost is spent on the central processor, as against more than 50% on software and engineering. The actual cost of a microprocessor is therefore only to a very limited extent determined merely by the hardware cost. Even so, the microprocessor is assured of a future in cement works, but more particularly as a powerful component unit of process control systems.

VI. Process control system

The components and subsystems described in the foregoing sections co-operate within the overall framework of a process control system. An important condition for reliable combined action and economical operation is to have properly interadjusted elements which are readily combinable with one another – also for subsequent extensions of the control system. Uniform interfacial structure and consistent signal language are especially important aspects to consider.
Besides, the following criteria should be applied in choosing a process control system:

The reliability of any process control system is determined by the quality of the elements it comprises and by the chosen structure of the system, which should be duly suited to the requirements of the process to be controlled and monitored. In the cement works the various sectors or stages of the process are "decoupled" from one another in a time context by silos, bins and storage buildings, each with a certain buffering capacity (Fig. 16), so that if they are of sufficiently ample design, the individual plant sections – crushing plant, raw mill, kiln, finish grinding mill, despatch facilities – can be operated independently of one another, within limits, of course. At the lowest level of safety functions a vertical structuring has proved

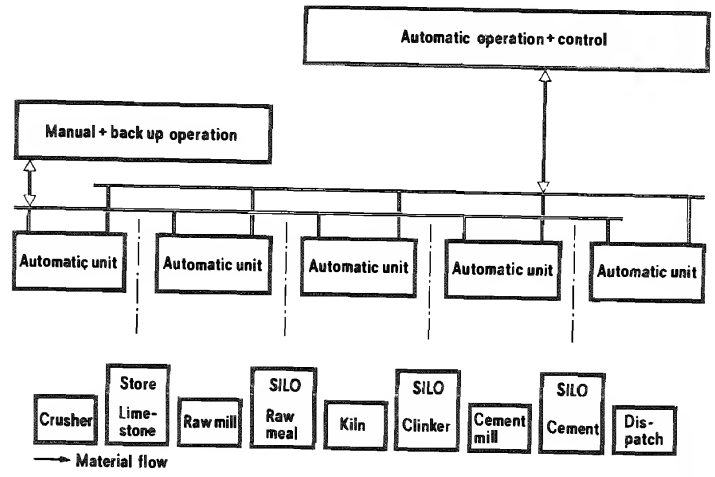

Fig. 16: Control system for a cement plant

advantageous for this plant configuration. To each part of the plant is assigned an automatic unit which is as self-sufficient as possible. If one such unit develops a fault, only one section of the plant will be affected in consequence. The other parts of the process can continue to function unaffected – for a time, anyway.
This bottom level accommodates all the functions that are essential to safeguarding the proper operation of the plant. The manual interventions in the process events are effected through this level. Programmable automation units are especially advantageous. Thanks to their flexibility, units which are alike in design can be adapted to perform different and wide-ranging duties. By thus using identical units of control apparatus for all sections of the plant the demands of spare parts inventory and servicing are substantially eased. Faults can be detected and put right more quickly, so that plant downtimes are reduced. Programmable systems are now available which can process binary and analog signals in the same equipment. It is thus possible to use one automation unit to perform all the following functions for a given section of the plant: measuring and process control (adaptation of analog quantities, pulse counting, limit monitoring, DDC control, calculation of characteristics), motor control and other open-loop functions (interlocking and sequencing, plant operation messages, alarm signal processing), and supervisory duties (signalling, logging, video display, fault locating). In addition, such a unit carries out the preparation and concentration of process data for transmission to higher-order levels for plant operation and process optimization.
The level for manual operation and back-up instrumentation comprises group control, group signalling and a few instruments for important process variables. The equipment employed on this level of functions is intentionally kept to the essential minimum for stabilized, but not optimized, controlled running of the plant.
"Comfortable" individual signalling and process optimization are achieved with the aid of the computer at the next higher level, the level of automatic functions. These functions require particularly efficient and therefore expensive system components. They are not, however, absolutely essential to steady-state operation. If a fault develops, the subordinate (lower) level for manual operation and back-up instrumentation will take over the necessary functions. For reasons of economy it is therefore possible to dispense with vertical structuring or redundancy of the optimization level. This approach is supported by developments in semiconductor technology, enabling more and more functions formerly performed at the optimization level to be transferred to the suitably strengthened and efficient bottom safety level.
Two basic forms of electrical interconnection of a process control system are possible. Fig. 17 shows the layout based on the radial cable principle. The signal cables coming from the plant converge in the control centre, where a process computer performs the functions of the automatic level. This layout has the advantage of clarity, while the cost of cabling is less than that for a conventional system, thanks to the combining of the signals in the automation units. A further reduction in the cost of cabling is possible by using the bus cable layout (Fig. 18), though it must be borne in mind that the cost of the actual bus system and of the

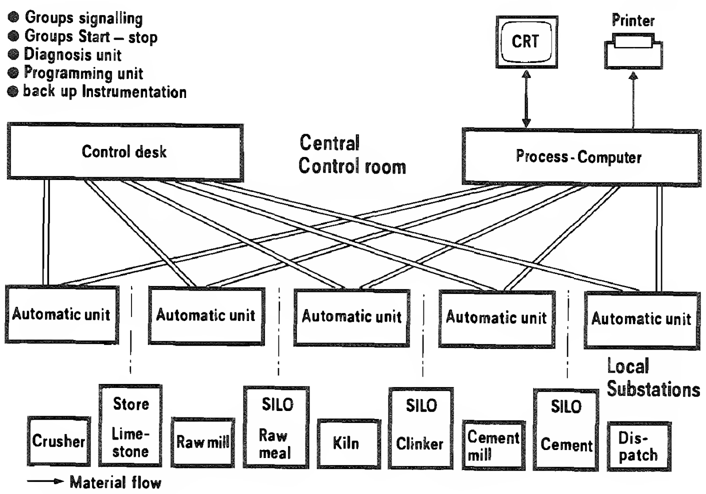

Fig. 17: Radial cable layout of a process control system

co-ordinator which may be needed in conjunction with it may wipe out the savings on cables and not really be justified by the supposed gain in reliability.

Other considerations with regard to reliability and economy arise with regard to erection, commissioning, maintenance and the correction of faults. During the service life of an installation it is more particularly the two last-mentioned points that become increasingly important in connection with the operational availability of the plant as a whole and the cost of keeping it in working order. This aspect is more particularly affected by the choice of system assembly technology, power supply, cabling layout and configuration of the connections within the system, as well as by considerations of documentation and facilities for fault detection and fault locating. If these viewpoints are not given due attention in the planning and design stage, plant operation is bound to suffer seriously in consequence (downtime, costs). With regard to documentation and fault correction it is important to remember that routine maintenance of the system will have to be carried out by ordinary electricians, not highly qualified electronics engineers!

In this context the training of the cement works' own technical personnel is of course very important. The works management should also be adequately acquainted with the principles and main features of the process control system. The suppliers of such equipment offer suitable training schemes which can be tailored to suit individual requirements. Instruction is given in special training

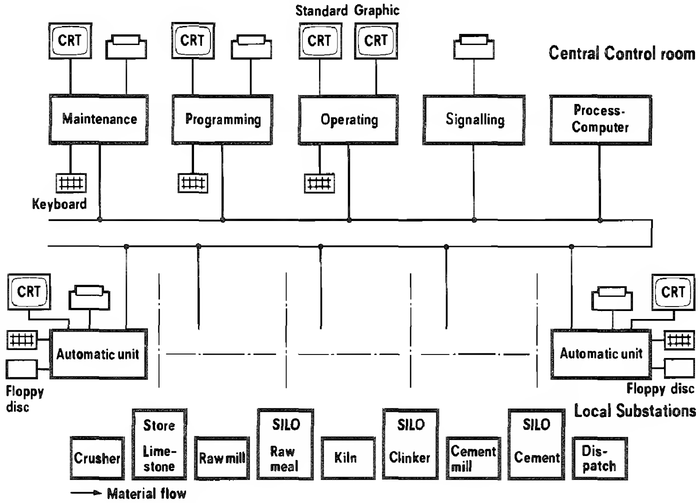

Fig. 18: Bus cable layout of a process control system

centres (e.g. the equipment manufacturer's factory or training centre) and may also be organised to tie in with the design, erection and commissioning of the actual system in the customer's works. Such training costs money, and it is advisable to make the necessary arrangements already at the time of signing the contract for the supply of the control equipment. Depending on the size and complexity of the control system, as well as on the scope of the task it has to perform (including possible future extension), these training programs and schemes for familiarizing the works personnel – repair and maintenance men, control room operators, plant engineers and cement works management – will have to be suitably adjusted to the needs of the occasion and to the level of training to be given to the different categories of personnel. The choice of the right type of training in any given instance is a matter of judgment, which can be assisted by test programs made available for the purpose by various equipment manufacturers.

The electric power supply to a process control system must be reliable and substantially unaffected by the rest of the power supply system in the cement works. There are two reasons for this requirement: for one thing, in the event of major faults in the works supply system (e.g., power failure), the monitoring of the various items of plant must continue to function, in order to ensure properly controlled stopping and subsequent restarting. In addition, electronic equipment is sensitive to fluctuations in the supply voltage, such as may occur due to the

direct switching-on of large motors. In present-day practice the more important items of electronic equipment powered from the mains are connected to standby systems which ensure continuity of power supply. More particularly, such systems comprise buffer batteries which can provide current to bridge any power cuts of limited duration (up to about 15 minutes). For reasons of economy, such arrangements should not be relied upon to cope with longer failures in the electricity supply. Under such circumstances it is better to switch to the cement works' standby generating system, i.e., the whole process control system should be connectable to it.

Taking advantage of the possibilities nowadays offered by electronic equipment, there is a tendency to specify high degrees of accuracy in performance (and correspondingly high complexity of the algorithms employed), which are hardly capable of fulfilment in actual plant operating practice. The old principle of the chain is applicable here: its strength is no greater than that of its weakest link, and a control system cannot be more accurate and dependable than the interconnection of its component parts allows.

In the cement industry there are limits to attainable accuracy imposed by the very nature of the materials handling and processing equipment employed, and not even the best process control system can improve on such limits. Accordingly, in designing the system it is of prime importance to suit the actual accuracy requirements to the conditions of the process itself and not go for the maximum that the makers of the process control equipment are prepared to claim or guarantee. In other words, a realistic approach is needed.

For instance, of what use is a mathematical kiln control model ensuring minimum energy costs if, to achieve its stated purpose, such a model requires undisturbed operation of the whole plant for periods of more than 24 hours?

Any practical engineer will know that in a cement works this condition is virtually impossible to fulfil.

A process control system designed on the assumption that unrealistic conditions can be fulfilled will be only of limited viability, and the money spent on it will yield a poor return.

With the further development of semiconductor technology the technical features of the process control systems used in cement works are bound to change and evolve. Their efficiency and performance capacity will increase and will thus contribute to the step-by-step solution of other specific problems associated with the production of cement. At the same time, however, the assessment criteria for successful use of such systems, as outlined in this chapter, will continue to be valid.

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H. Environmental protection and industrial safety

By G. Funke

I. Environmental protection

1 Prevention of air pollution

1.1 Dust-type emissions

The production of 1 tonne of cement involves the comminution of about 2.6 to 2.8 tonnes of raw materials, clinker, gypsum, blastfurnace slag, trass and (in coal-fired systems) coal to dust-like fineness. Between 5 and 10% of these finely pulverized materials will be agitated and thus suspended as dust in gases and will have to be substantially removed from these before discharge into the atmosphere. Depending on plant operating conditions, the quantity of gas or air to be dedusted per kg of cement production is between 6 and 12 m^3.

The "dust" arising in the various processing units of a cement works varies greatly in composition. In the main, the following types of dust are to be distinguished:

raw material dust (e.g., from limestone, lime marl, clay, iron ore, gypsum, blastfurnace slag);
raw meal dust;
cement kiln dust (exit gas dust);
clinker dust;
coal dust;
cement dust.

Table 1: Chemical composition of cement kiln dusts before and after the dust collector

	rotary kiln with cyclone preheater (without exit gas utilization)		rotary kiln with grate preheater	
	before collector	after collector	before collector	after collector
SiO_2	10 to 18	7 to 11	6 to 22	2 to 19
$Al_2O_3 + TiO_2$	3 to 9	3 to 6	1 to 13	0.5 to 8
$Fe_2O_3 + Mn_2O_3$	1 to 4	1 to 3	0.5 to 5	0.5 to 4
CaO	39 to 47	41 to 51	12 to 47	6 to 26
MgO	0.5 to 2	0.5 to 2	0.5 to 3	to 2
K_2O	0.5 to 3	0.5 to 4	3 to 40	14 to 40
Na_2O	to 0.2	to 0.5	0.5 to 5	0.5 to 3
SO_3	0.5 to 2	0.5 to 4	6 to 30	7 to 41
F^-	n.b.	0.10 to 0.13	0.05 to 0.25	0.03 to 0.25
Cl^-	to 0.5	to 0.3	0.5 to 20	0.9 to 4.5
loss on ignition ($CO_2 + H_2O$)	29 to 38	29 to 38	7 to 20	4 to 24

Table 2: Proportion (by mass) of particles under 10 microns in size in dust-laden gases

	percentages by mass
crusher for limestone	5–20
rotary dryer for raw material	40–70
rapid dryer for raw material and coal	50–70
grinding/drying plant	40–90
rotary kiln with cyclone preheater	85–99.5
rotary kiln with grate preheater	10–45
shaft kiln	15–30
grate cooler	0–15
tube mill (cement, raw meal, coal)	40–80
material handling devices, packing machines, silos	10–50

With the exception of kiln dust, the above-mentioned dusts are of the same composition as the material from which they initially arise. **Kiln dust**, i.e., the dust carried out of the kiln with the exit gas, consists of thermally unchanged raw meal, dehydrated clay, decarbonated (calcined) limestone, and newly formed minerals corresponding to all stages of processing up to the clinker minerals; besides, in coal-fired kilns, it will contain ash constituents from the fuel. Table 1 gives the chemical composition of the dusts discharged from two types of kiln. The values encountered in the dust before and after the dust collecting equipment, respectively, are listed. The dust constituents are present mainly as carbonates, silicates, sulphates and chlorides. Alkali, sulphur and chlorine compounds are found to be more particularly concentrated in the cleaned gas dust, these compounds having been volatilized in the burning zone of the kiln.
About 60 to 70% of the coal ash is absorbed into the clinker, while the remainder of this ash is discharged as kiln dust.
Table 2 gives some guiding values for the percentages by mass (or weight) in the dust particles below 10 microns size, which are extracted from the various dust sources envisaged here and are precipitated in the dust collecting equipment, in some cases after having passed through a pre-cleaner.
In the cleaned gas discharged from the collecting equipment the percentage of particles below 10 microns is between 80 and 90, for dust content values which are within the upper limits allowed by the German pollution prevention regulations. These are embodied in "Technische Anleitung zur Reinhaltung der Luft" ("Technical directives for clean air").

1.2 Gaseous emissions

The exit gases of cement kilns consist substantially of nitrogen N_2, carbon dioxide CO_2, oxygen O_2 and water vapour H_2O. In addition, they may contain small

amounts of sulphur compounds (SO_2) and nitrogen oxides (NO, NO_2), as well as carbon monoxide CO and hydrogen sulphide H_2S.
Sulphur contained in the raw materials and the fuel is oxidized to sulphur dioxide SO_2 at temperatures above 1000° C in the presence of excess air. This compound reacts with the alkalies that volatilize at the same time and forms alkali sulphates, which possess low volatility and are discharged from the kiln with the clinker or the dust. The residual SO_2 in the kiln gas can, in the presence of oxygen, react with the $CaCO_3$ of the feed material and also with the CaO (formed from this material by heating) to give calcium sulphate $CaSO_4$. This reaction is promoted more particularly in drying/grinding plants and in conditioning towers by the grinding process and the presence of water vapour.
If there is an excess of alkali, a high proportion (88 to 100%) of the total sulphur introduced into the kiln system is combined in the cement clinker and the kiln dust. Only the remaining small proportion (less than 12%) is emitted as SO_2 with the cleaned gas. In the event of sulphur excess, the SO_2 emission is liable to be higher (see Locher/Sprung/Opitz, 1971).
Carbon monoxide and **hydrogen sulphide** are formed only under conditions of incomplete combustion, e.g., in shaft kilns, and even then only in small amounts.
With air excess, **nitrogen oxides** may form in cement kilns, namely, nitrogen monoxide NO and nitrogen dioxide NO_2 in the volumetric ratio of about 90% NO to 10% NO_2. The content of nitrogen oxides in the exit gas is between about 200 and 1100 mg/m^3 (corresponding to 150–800 ppm), reckoned as NO. The values at the lower end of the range are more particularly valid for kilns with precalcining.
Gaseous **chlorides** and **fluorides** do not occur in the kiln exit gases, since the very small amounts of chloride and fluoride contained in the raw materials combine with the alkalies and the calcium in the clinker and dust in the course of the cyclic processes in the cement kiln.
If **grinding** aids are used in the grinding of cement clinker, a substantial proportion of these organic compounds is combined in the cement, while the remainder is emitted in vapour form. No precise information on the actual amounts of such vapour is as yet available, however.

1.3 Exit gas conditions in cement works

Table 3 gives information on the dust and exit gas conditions of the cement production plants currently operated in the Federal Republic of Germany. These details are necessary for guidance in determining the capacity of dust collection equipment to be installed. The specific exit gas volumes in m^3 under standard conditions (0° C, 1013 mbar pressure) comprise the water vapour arising from the drying process in the plant, the firing unit supplying hot air for drying, and the additional water injected for conditioning the exit gases. The exit gas analysis (CO_2, O_2, CO) is referred to dry gas. The water content of the exit gases can be calculated from the dew-point. The dust content of the gas (i.e., the dust-laden gas before cleaning) is expressed in g per m^3 under standard conditions (moist)

Table 3: Exit gas conditions of cement production plants

	long wet-process kiln	rotary kiln with cyclone preheater (without exit gas utilization)	rotary kiln with grate preheater	shaft kiln	grate cooler	rotary dryer	rapid dryer	grinding/ drying plant	tube mill
feed	raw slurry	raw meal	pellets	pellets	clinker	raw material	raw material, coal	raw material, coal	raw material, clinker, gyspum, blast-furnace slag
water content of feed [% by weight]	32–40	0.5–1.0	11–22	8–14	–	2–15	2–15	2–10	0–2
spec. exit gas volume [m^3/kg[1)]]	3.2–4.2	1.5–1.8	1.8–2.2	2.0–2.8	0.7–1.8	0.8–2.0	0.5–1.5	0.8–1.5	0.2–0.8
exit gas analysis									
CO_2 [%]	25–18	33–20	29–20	26–10	–	<4.0[3)]	<4.0[3)]	<4.0[3)]	–
O_2 [%]	4– 8	3– 9	4–10	5–10	21	18–20	18–20	18–20	21
CO [%]	<0.1	<0.1	<0.1	4– 1	–	–	–	–	–
exit gas temperature [°C]	130–180	280–360	90–150	45–125	200–400	70–150	70–150	70–150	60–120
dew-point [°C]	65–75	35–45[2)]	50–65	40–44	<25[2)]	40–70	40–70	40–60	20–60
dust content of gas [g/m^3[1)]]	2.0–15.0	15–40	1.0–4.0	1.5–5.0	0.7–10.0	20–60	20–150	30–800	30–400
dust precip.	electr. precip.	electr. precip.	electr. precip.	electr precip.	electr. precip. fabric filter granular bed filter	electr. precip. fabric filter	electr. precip. fabric filter	electr. precip. fabric filter	electr. precip. fabric filter

1) referred to volume flow rate of gas under standard conditions (moist)
2) without additional water spraying
3) with own air heating unit
4) obsolete in the Fed. Rep. of Germany

upstream of the dust collecting equipment. The dust collectors mentioned in the table are of the types currently employed in Germany.

1.4 Influences upon the dust loading

The dust content of the dust-laden gas depends substantially on the nature and fineness of the material and the velocity of the gas flow with which it comes into contact.

In **long wet-process kilns** the exit gas dust content is affected more particularly by the filtering or dust arresting action of the internal fittings (chains, special ceramic inserts). If the kiln output is increased in conjunction with higher gas flow rates, changes in the internal fittings and higher exit gas temperatures, the quantities of dust produced will correspondingly increase.

The dust content of the gas discharged from the preheater of a **suspension preheater kiln** is mainly governed by the dust collecting action of the top cyclone stage, its operational efficiency and the tightness of closure of its discharge locks.

The dust content of the exit gas from **kilns with grate preheaters** (Lepol kilns) is substantially dependent on the character and stability of the pellets and on the filtering action of the bed of pellets on the grate in the drying chamber. The dust collecting action of the intermediate gas cyclones can reduce the amount of dust arising in the drying chamber.

The amounts of dust contained in the **gases from shaft kilns** depend on the filtering action of the layer of moist pellets located above the burning zone. Irregular kiln operation with fire "break-outs", e.g., in consequence of poor pelletizing and irregular raw meal and fuel proportioning, will increase the amounts of dust produced, as will also overloading of the kiln.

The dust content of the **exhaust air from grate coolers** is affected by the granulometric composition, the degree of burning and the bulk density of the clinker and also by the cooling air flow rate employed. A clinker breaker after the cooler may increase the dust burden. The dust content is especially high in the event of underburning and if flushing (sudden rushing) of raw meal occurs.

In **rotary dryers** the internal fittings do indeed improve the transfer of heat between the hot gases and the material being dried, but they also throw up considerable amounts of dust, especially with friable materials. High values of the dust content in the exhaust gases are liable to occur more particularly with **rapid dryers** as a result of the comminuting action of the shaft with flights revolving at high speed.

The dust content in the exhaust air from **grinding mills** (roller mills, tube mills, aerofall mills) is strongly dependent on the mill system concerned, the feed material, the size reduction effected (product fineness), the exhaust air flow rate and especially the air flow velocity in the mill and air exit passages. Air-swept tube mills and roller mills have a very heavy dust burden in their exhaust air, since all the finished product of the grinding process is carried along in this air and has to be precipitated from it.

The amounts of dust produced in **crushers** will depend on the crusher design features, the nature of the feed material, its grading and its moisture content. If the

feed material is very moist (4–8% water or more), no arrangements for dedusting the exhaust air will generally be necessary, as little or no dust is thrown up. Besides, dust production can be reduced by spraying water into the crusher.

Gentle **handling** of dry bulk materials on conveyors, elevators, etc. will always help to cut down the amounts of dust produced. More particularly, this approach to the dust problem in materials handling will include the use of handling devices which develop little abrasive action and the avoidance of large heights of fall of the materials. Suggestions to that effect are given, for example, in the VDI 2262 Code entitled "Combating dust nuisance at the place of work".

In general, any operation involving the suction of air through a flow of material should be avoided, if possible, because this substantially increases the dust content of the exhaust air (classifying effect). When materials are deposited onto an outdoor stockpile, preliminary removal of the finer particles may be advantageous as a means of curbing dust nuisance.

1.5 Extracting the dust

The dust thrown up in the operation of the various parts of the cement works is extracted by suction devices and removed from the exhaust gas or air flow in separators of various kinds.

Dust extraction devices comprise exhaust hoods, pick-up nozzles, etc. Suggestions for the design of such devices, together with guiding values for the air flow velocities required, are given in above-mentioned VDI-2262 Code.

Information on exhaust air or gas flow and dust content from the main items of cement production plant (kilns, coolers, dryers, mills) is given in Table 3. The dust content and required air flow rates for dust extraction from other cement works equipment are given in Table 4, based on information supplied by manufacturers of dust collecting systems. The flow rates indicated in the table, related to the operating condition of the gas, depend to a great extent on the efficiency of the extractor systems, on the nature and fineness of the material, and on the size and throughput rates of the installations in question. Hence these values vary within wide ranges. In general, if a particular unit of machinery or other equipment is enclosed in a casing so that cross-flows are reduced as much as possible, the amounts of air to be extracted for dedusting are considerably reduced.

For **crushers** the exhaust air flow is roughly 60 m^3/min per m^2 of the opening of the feed housing (see Duda, 1978). The exhaust air from **screening installations** can be estimated at 15 m^3/min per m^2 of screen surface plus 30 m^3/min per m^2 of outlet and other openings. For **bins** the extraction air flow is sometimes put at 75 m^3/min per m^2 of bin cross-section. In the case of **receiving hoppers** into which dry and fine-grained bulk materials are tipped, a multiple of the air volume displaced by the tipping or dumping operation must be extracted at suitable points, while the receiving opening of the hopper should be as small as possible and closed as effectively as possible by means of rubber aprons or the like.

Transfer points of belt conveyors for dry bulk materials containing appreciable amounts of fine particles require air extraction rates estimated at 60 m^3/min per m^2

Table 4: Dust content and flow rates of exhaust air from other cement works installations

	dust content of exhaust air	flow rate of exhaust air to be dedusted	
roll crushers	0.5 – 2.0	60 – 80 m^3/h per t of product	
hammer crushers	5 – 15	80 – 100 m^3/h per t of product	
impact mills	10 – 20	150 – 200 m^3/h per t of product	
screening machines	5 – 20	500 – 1200 m^3/h per m^2	
bucket elevators < 1 m/s	5 – 30	2000 m^3/h per m^2	of bucket elevator cross-section
bucket elevators > 1 m/s	5 – 30	2800 m^3/ per m^2	
transfer points of belt conveyors	5 – 20		
width 600 – 800 mm		1500 – 2400 m^3/h	for feed and
1000 – 1200 mm		2100 – 3000 m^3/h	discharge
1400 – 1600 mm		2400 – 3600 m^3/h	respectively
pneumatic trough conveyors	30 – 50	120 m^3/per m^2	of trough area
		+ 20%	for cold
		+ 30 – 35%	for hot material
pneumatic conveyors	150 – 200	conveying air volume	
airlift		+ 100%	
Fuller pump		+ 50%	
pressure vessel		+ 200 – 400%	
silo installations	5 – 15		
feed pneumatic		same as for pneumatic conveyors	
feed mechanical		material quantity in $m^3/h \times 3.5$	
pneumatic homogenization		air volume for aeration + 30 – 40%	
cement and clinker loading for despatch	10 – 60		
road vehicle		ca 3500 m^3/h	approx. according to type and
railway waggon		ca. 3500 m^3/h	size of loading
barge or ship		10000 m^3/h	attachment
cement sack packing machines	5 – 30	2000 m^3/h	per spout
sack cleaning machines	2 – 5	2000 – 3000 m^3/h	

	dust content of exhaust air	flow rate of exhaust air to be dedusted
dumping from road or rail vehicles into hoppers	5 – 20	1800 – 2000 m^3/h per m^2 of grizzly area, or material quantity in $m^3 \times \frac{3600}{5}$ corresponding to volume of air displaced by material in 5 sec

of the open intake cross-sectional area of the exhaust hood. Higher rates may have to be adopted for belts running at high speeds.

The air extraction rate per minute for dust removal from **cement bins** and similar vessels associated with bulk cement handling should be approximately three times the bin volume.

For **cement packing machines** (sack fillers) with rotary impellers the air extraction rate is about 35 m^3/min per filling spout. To this should be added about 3 m^3/min per spout for the feed bin over the packing machine.

Air extraction rates for cement sack cleaning installations over belt conveyors are roughly 500 m^3/min.

1.6 Handling the dust

1.6.1 Pipelines

Pipelines and ducts for conveying the dust extracted from the various items of plant should be so dimensioned that no dust will be deposited from the air in which it is carried along. Horizontal pipes should be avoided as much as possible. Where the pipeline has to run horizontally, the average gas or air velocity in it should be between 16 and 22 m/sec for a dust content of up to about 50 g/m^3. For higher values of the dust loading it is advisable to increase the velocity.

If abrasive dust is to be handled, bends and fittings (e. g., branch pieces) should have thicker walls and/or be lined with special wearing plates or with wear-resistant ceramic materials. In addition, the pipelines should be provided with cleaning openings which should be properly accessible and tightly closable. Also, for each extraction point there should be a control valve in the pipeline for adjustment of the flow rate. If moist gases have to be conveyed, the pipelines should be suitably **insulated** and, if necessary, be additionally fed with warm dry air in order to prevent the temperature falling below the dew-point, because the resulting condensation moisture is liable to cause agglomeration of dust particles and choking of the pipeline.

Once the location of the extraction points and of the dust collecting equipment (precipitator, filter) has been determined and the air or gas extraction flow rates

have been estimated, the system of pipelines or ducts for conveying the dust can be designed. It is advisable not to use very long pipelines for the sake of, for example, connecting a large number of extraction points to one and the same central dust collection unit. Quite often it is more advantageous to install a number of small **individual filters** in the vicinity of the actual dust extraction points. This will usually be associated with lower power consumption because the pressure drop in the shorter pipelines is less. Besides, small filters offer operational and servicing advantages.
If a volume flow rate $\dot{V}$ (in m^3/sec) with average gas or air velocity $\dot{w}$ (in m/sec) is required, the cross-sectional area of the pipeline F (in m^2) will have to be: $F = \dot{V}/\dot{w}$.

1.6.2 Fans

For moving the dust-laden air or gas through a pipeline it will be necessary to install a suitable **fan**, which will have to develop the required flow rate, while maintaining the pressure difference at the extraction points and in the pipeline system as well as in the dust collection equipment itself and in any further pipes or ducts downstream thereof. The total pressure difference Δp is the sum of the static and dynamic pressure differences and is often called the overall pressure rise developed by the fan.
The **power** p (in kW) that a fan must develop is, in the range of small relative pressure rises, proportional to the overall pressure rise Δp (in N/m^2) and to the volumetric flow rate $\dot{V}$ (in m^3/sec):

$$P = \dot{V} \cdot \Delta p/1000.$$

The actual power consumption of the fan at the shaft is:

$$P_w = \dot{V} \cdot \Delta p/1000 \cdot \eta,$$

where η is the **efficiency** of the fan: it depends on the internal mechanical losses and is defined as the ratio of the fan power rating P to the power input at the drive shaft P_w. In practice, the fan efficiency generally has a value in the range between 0.65 and 0.8.
The performance of a fan is characterized by its **characteristic curve** and represents the relation between the overall pressure rise and the inlet volume flow rate for a certain speed of the fan. It is known also as the pressure-volume curve. The appropriate characteristic chart showing these curves for various speeds, as well as the power consumption and the efficiency as functions of the flow rate, should be available with every fan.
The **pipeline characteristic** (resistance characteristic) is the relation between the volume flow rate and the pressure drop in the pipeline system. The intersection of the fan characteristic and the resistance characteristic is the **operating point** of the fan, which is automatically achieved by the fan in service. The resistance characteristic can be varied by closing or opening a control valve.
The **pressure drop** in the pipeline system is caused more particularly by frictional losses between the moving gas or air and the stationary wall of the pipe. The frictional loss Δp_R is expressed by:

$$\Delta p_R = \lambda \cdot \frac{L}{D} \cdot \frac{\varrho}{2} \cdot w^2 = \lambda \cdot \frac{L}{D} \; p_d,$$

where:

λ	=	pipe friction coefficient
L	=	length of pipeline in m
D	=	diameter of pipeline in m
ϱ	=	density of the medium in kg/m^3
w	=	flow velocity in m/sec
p	=	$\frac{\varrho}{2} \cdot w^2$ = dynamic pressure in N/m^2
Δp_R	=	frictional loss in pipeline in N/m^2.

The friction coefficient λ is dependent on the Reynolds number and the absolute roughness of the pipe wall. Fig. 1 shows λ as a function of the volumetric flow rate $\dot{V}$ for various roughness values.

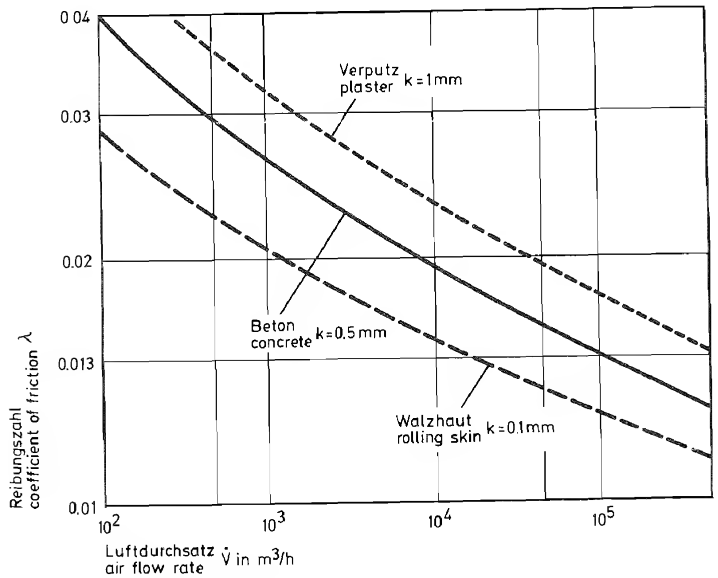


Fig. 1: Coefficient of friction λ as a function of volume flow rate

Table 5: Resistance coefficients of pipe fittings

type of fitting flow pattern	resistance coefficient	remark
round inlet	0.5	
square inlet	0.7	
round inlet with tapered portion	0.3 to 0.5	according to shape of taper
pipe bend		
R/D = 1.0	0.5	R = radius of curvature
R/D = 1.5	0.4	D = pipe diameter
R/D = 2.0	0.27	
R/D = 3.0	0.2	
right-angled and skew tees		
$\alpha = 20°$	0.15	α = angle between centre-lines of intersecting pipes
$\alpha = 30°$	0.2	
$\alpha = 45°$	0.3	
$\alpha = 60°$	0.5	
$\alpha = 90°$	1.0	
gradual widening of cross-section		
$\alpha = 20°$	0.4	α = angle of taper,
$\alpha = 40°$	0.8 to 0.95	for diameter ratios
$\alpha = 60°$	1.0 to 1.2	$\frac{D_2}{D_1}$ from 1.5 to 3
$\alpha = 80°$	1.03 to 1.16	
$\alpha = 100°$	1.03 to 1.1	
discharge opening (chimney)		
deflector	1.2	
cowl	1.4	

Besides frictional losses, other losses in pipelines are caused by resistance due to bends, internal features, changes in pipe cross-section, branches, junctions, etc. The pressure losses due to these various features can be calculated from the following general expression:

$$\Delta p = \zeta \cdot \frac{\varrho}{2} \cdot w^2$$

where:

ζ = resistance coefficient
ϱ = density of the medium in kg/m^3
w = flow velocity in m/sec.

Values of the resistance coefficient for various pipe fittings and other features are given in Table 5.

1.7 Measures for the reduction of dust emission

1.7.1 Separator systems

The various dust separators used in the cement industry can be broadly subdivided into two categories: separators within the production process (for dust removal from the air or gas discharged from grinding plants, preheaters or pneumatic conveyor equipment) and separators for dust removal from the air or gas discharged into the atmosphere. The dedusting devices of the first category are mostly inertia-force separators, sometimes electrostatic precipitators. On the other hand, inertia-force separators are seldom used for the prevention of atmospheric pollution; for this purpose the two general types almost exclusively employed are electrostatic precipitators and filters (fabric filters, granular bed filters). Wet collectors (scrubbers) are hardly used in connection with cement manufacture.

1.7.1.1 Inertia-force separators

The following types are to be distinguished within this general category of separators (dust collectors) (see VDI 3676):

- **Counter-current gravity separators** (dust settling chambers with vertical gas flow): the particles are precipitated by the action of gravity, i.e., they descend in the rising stream of gas, which is thus relieved of its dust burden (Fig. 2).
- **Cross-current gravity separators** (dust settling chambers with horizontal gas flow): the particles are precipitated by the action of gravity, here directed transversely to the gas flow direction. This type of separator with gas velocities of 0.5 – 1.5 m/sec used to be widely employed as primary dust collectors (precleaners) (Fig. 3).
- **Inertial separators** involving changes of flow direction of the gas stream, which is thus relieved of dust because the particles, due to their inertia, are not able to follow the gas flow path (Fig. 4). In certain types of separator the dust-laden gas impinges on baffles or other bodies and, in being deflected around these, loses its dust particles because of their greater inertia. Such dust collectors are more particularly called **impingement separators.**
- **Cyclone separators** (Fig. 5): these rely on the action of centrifugal forces on the dust particles carried along in the swirling stream of gas. The particles are thus flung radially outwards to the wall of the cyclone, from where they fall into the dust hopper. The centrifugal force which determines the collection efficiently is directly proportional to the mass of the particles and to the square of the circumferential velocity, but inversely proportional to the radius of the cyclone:

  centrifugal force $Z = m \cdot u^2/r$.

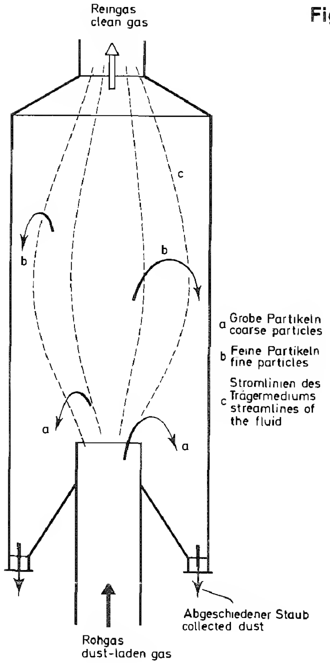


Fig. 2: Counter-current gravity separator

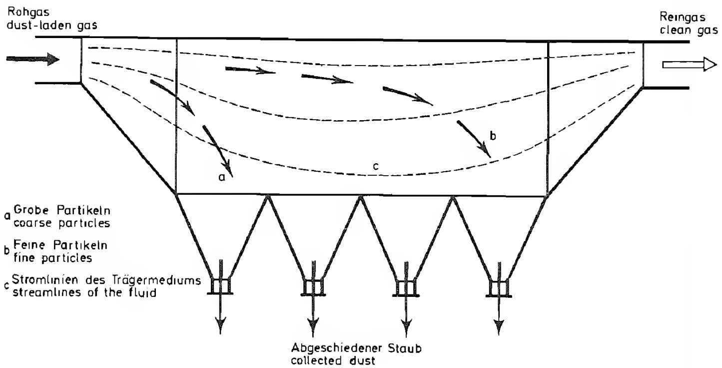


Fig. 3: Cross-current gravity separator

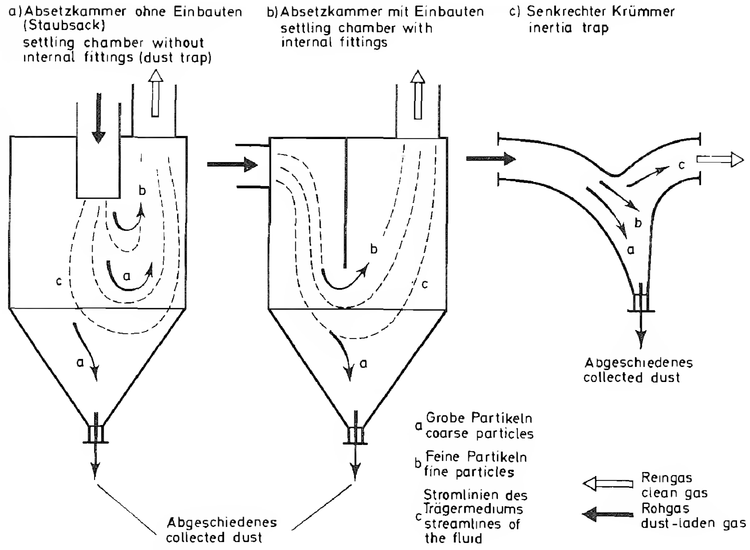


Fig. 4: Inertal separators

In the cement industry, cyclones are used mostly as separators within the production process and as pre-cleaners for high-efficiency dust collectors in cases where air or gas with high dust loadings has to be treated; they are used as dedusting devices only for gases with a low content of dust which moreover consists of relatively coarse particles, e.g., the exhaust air from the clinker coolers of Lepol kilns.

Various arrangements of cyclones are to be distinguished

- **individual cyclones:** single units of large diameter (above 1500 mm) and height);
- **multiple cyclones:** batteries of cyclones connected in parallel, the gas flow being distributed as uniformly as possible to them; diameters are in the range of about 500 to 1500 mm; see Fig. 6;
- **multi-cyclones,** generally below 500 mm diameter, arranged in banks, as shown in Fig. 7, and provided with a common air inlet and dust hopper system.

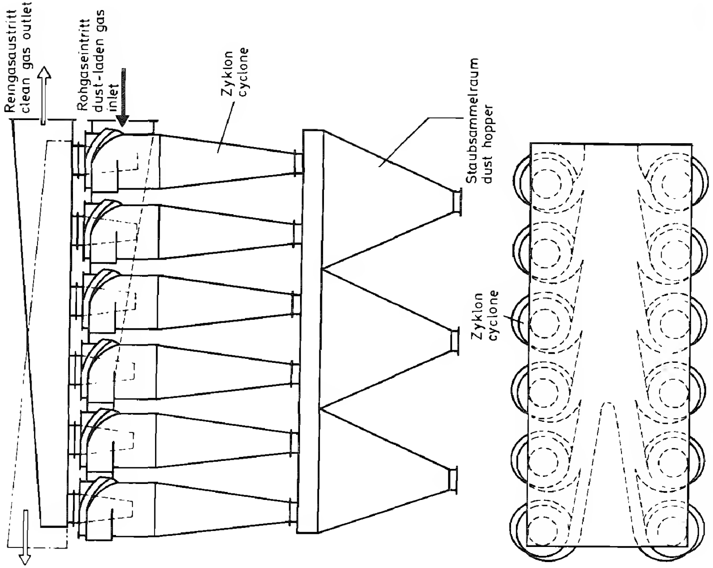


Fig. 6: Multiple cyclone

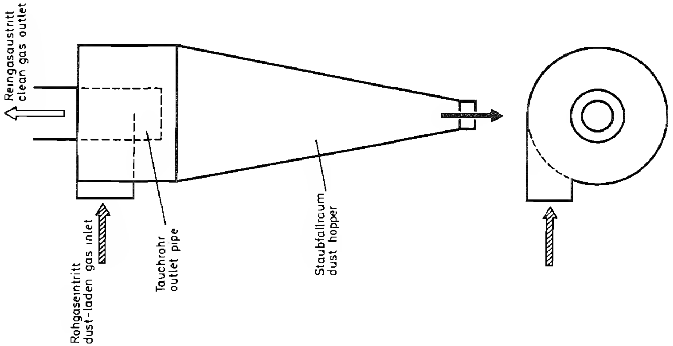


Fig. 5: Cyclone separator

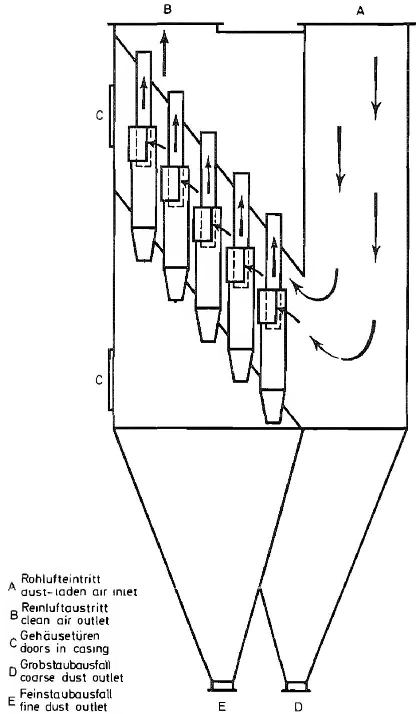


Fig. 7: Multi-cyclone collector

General features of cyclone separators are as follows:

- diameter of cylindrical part: D = 200 to 3500 mm;
- overall height of the separator: 3 to 5 D;
- cross-sectional rating (resultant flow velocity in the cross-section of the cylindrical part): 1 to 3.5 m/sec;
- inlet velocity: 10 to 24 m/sec;
- exit velocity in outlet pipe: 8 to 13.5 m/sec;
- pressure drop through cyclone: 5 to 15 mbar;
- operating temperature: up to 500° C.

The pressure drop Δp (in mbar) is approximately proportional to the square of the gas velocity (in m/sec):

$$\Delta p = \zeta \cdot \frac{\varrho}{2} \cdot w^2.$$

where ϱ is the density of the gas (in kg/m^3) and ζ is a pressure loss coefficient (for example, $\zeta = 0.4$).

The **overall collection efficiency** is affected by the selective separation of the dust according to particle size in accordance with a separation curve. For constant gas flow rate, the overall collection efficiency increases if the following influencing quantities increase: settling velocity of the dust particles, agglomerating tendency of the dust, dust content of the gas to be dedusted (within limits).

The proportion of each particle size fraction which is precipitated in a cyclone of particular shape and rating is called the **fractional dust collection efficiency** and is plotted as a percentage against the particle size diameter (in microns). The curve thus obtained is called a separation curve. It is an important criterion for comparing different cyclones with one another. Table 6 indicates, by way of example, the overall collecting performance of a cyclone for dust with known particle size distribution.

Table 6: Dust collecting performance of a cyclone for dust of known particle size distribution (example)

particle size	dust to be collected % by weight	fractional dust collection efficiency in %	overall collection efficiency in %
0 to 5	5	60	3.00
5 to 10	10	85	8.50
10 to 20	30	95	28.50
20 to 30	17	98	16.66
30 to 40	13	99.5	12.94
40 to 50	7	99.9	6.99
> 50	18	100	18.00
	100		94.59

For any particular cyclone of given shape and rating there is a certain particle size of which 50% is precipitated, while the other 50% remains in suspension in the air and is carried out of the cyclone. This is called the cut size and likewise constitutes a criterion of cyclone performance.

Multi-cyclones occupy less space on plan than multiple cyclones of equal performance. Besides, because of their small diameter the individual cyclones achieve more powerful centrifugal action and thus attain higher efficiencies. In practice, however, there are considerable disadvantages to be set against these advantages:

- there is a high risk of blockage of the small cyclones;
- it is difficult to achieve uniform distribution of gas to the individual cyclones;
- the common dust collecting hopper is liable to cause "short-circuit" gas currents from cyclone to cyclone in the event of pressure differences at their respective outlets;
- effective sealing of the cyclones between the inlet chamber (into which the dust-laden gas is admitted) and the outlet chamber (which receives the cleaned gas) is difficult, especially in large installations, so that coarse dust particles are liable to get into the cleaned gas.

The drawback of gas short-circuit can be reduced by the extraction of gas from the dust collecting hopper.

Cyclones achieve their optimum separation or collection efficiency only if the dust-laden gas is admitted to them at the appropriate rate of flow on which the cyclone design was based. If the flow rate falls below this value, so that the cyclone is operated at too low a load, the collection efficiency goes down; on the other hand, if the cyclone is overloaded, the pressure drop and the amount of wear will increase.

Wear-resistant linings may take the form of wearing plates or ceramic materials, but must not adversely affect the gas flow pattern in the cyclone. Fine-grained dust, especially if it has a high alkali content, is prone to cause caking and choking. Insulation may be required in order to prevent the temperature in the cyclone falling below the dew-point.

1.7.1.2 Fabric filters

Fabric filters are extensively used in cement works for cleaning the exhaust air from tube mills, roller mills, dryers, crushers, screening installations, material handling installations, silos, bins and despatch loading plants. In conjunction with air coolers for lowering the temperature of the dust-laden air admitted to them, such filters can also cope with the exhaust air from grate-type clinker coolers. On the other hand, fabric filters are not used for dedusting the exit gases of cement kilns, at least not in the Federal Republic of Germany.

Dust precipitation in textile filter media is accomplished by the following processes:

- interception: the fibres of the filter medium act as a sieve or strainer;
- inertia: the gas flow is deflected around the fibres, while the dust particles are precipitated by virtue of their inertia;
- diffusion and electrical forces: these are significant only for the very smallest particles.

It has hitherto not proved possible to calculate accurately the **collection efficiency** of a technical filter medium. It is mainly a function of the porosity and the thickness of the medium, the fibre diameter and the collection efficiency of the individual fibre.

Among others, the following theoretical equation is given in the literature (see VDI 3677):

$$\eta_{ov} = 1 - e^{-\frac{1-\varepsilon p_o}{\varepsilon p_o} \cdot \frac{4}{\pi} \cdot \frac{L}{d_f}}$$

where:

η_{ov} = overall collection efficiency
εp_o = porosity in cm^3 of voids per cm^3 of filter material
L = layer thickness in m
d_f = fibre diameter in m.

Therefore the collection efficiency increases if:

- the pore ratio becomes lower, e.g., in consequence of dust that collects in the voids (plugging of the pores);
- the layer thickness of the filter medium increases;
- the fibre diameter decreases.

In selecting a suitable filter medium the aim will therefore be to have low permeability to air in combination with a high weight per unit area if high dust collection performance is to be attained. The effect of the dust deposited on or in the filter medium, and the periodic removal of part of this dust by cleaning action, must also be considered.
Besides collection efficiency, the **pressure drop** (corresponding to the flow **resistance**) is an important characteristic criterion of a filter medium. It can be expressed as follows:

$$\Delta p_F = K \cdot w_o \cdot \varrho \cdot \nu \cdot L,$$

where:

K = constant for the filter medium
w_o = flow velocity of gas admitted to filter medium in m/sec
ϱ = density of the gas in kg/m^3
ν = kinematic viscosity of the gas in m^2/sec
L = layer thickness of filter medium in m.

Besides the pressure drop through the filter medium there is the pressure drop due to the filter casing, which is calculated from the following formula (similar to that for pipelines):

$$\Delta p_G = \zeta_F \cdot \frac{\varrho}{2} \cdot w_1^2$$

where:

ζ = resistance coefficient
ϱ = density of the gas in kg/m^3
w_1 = velocity of entry into filter in m/sec.

When a **dust layer** has formed on the filter medium, this dust too will cause a pressure drop which is therefore additional to that of the medium itself.
The pressure drop in **woven-fabric filters** (cloth filters) is between about 8 and 20 mbar. For sizing the fan equipment it is necessary also to take account of the pressure drop in the pipelines and dust extraction devices (exhaust hoods, etc.), so that the pressure rise to be developed by the fans will generally range from 20 to 50 mbar.
Two main types of fabric filter media (fibrous media) are to be distinguished:

- woven fabrics comprising threads arranged in a rectangular mesh of warp and weft, in various weave patterns such as plain weave, twill weave or satin weave;
- non-woven fabrics, more particularly those formed and strengthened by mechanical action, e.g., by means of needles (needle felts), or with adhesives.

Felt-fabric filter media (more particularly **needle felts**) have gained in importance in recent years. In the "needling" process the fibres are intimately matted together by the action of numerous needles with barbed hooks. These felts are often provided with a woven fabric reinforcement to give additional strength and dimensional stability to the felt. Thanks to their closely matted fibre texture with approximately uniform pore structure, felts attain higher collection efficiency in conjunction with lower pressure drop than woven filter fabrics.
Important characteristic criteria of woven and non-woven fabric filter media are their weight per unit area, air permeability and thickness (Table 7).
Formerly, fabrics for filter media were made chiefly of wool, cotton or linen, but man made fibres are now increasingly used for woven-fabric as well as for felt-fabric media.
The physical properties of the most important fibrous filter materials used in the cement industry are listed in Table 8.
The ranges of application of the various materials are determined by their properties. In the cement industry, **needle felts of man-made fibres** are extensively used, more particularly polyester for the filtration of dry warm exhaust air, polyacrylonitrile for moist warm exhaust air (e.g., discharged from grinding/drying plants), and polyamide (Nylon, Perlon) for cold air carrying abrasive dust. Cotton, which is relatively inexpensive, is still used for the filtration of cold air containing dust with low abrasive action. Though wool has excellent filtering properties, it has the drawbacks of absorbing much moisture and having very limited temperature resistance. It is now little used. Nomex is occasionally used for filters subjected to high temperatures, but is relatively expensive. Teflon and glass fibres are materials which have hitherto not been used as filter media in the cement industry in the Federal Republic of Germany.
The **filtering properties** of fabric filter media can be improved by suitable mechanical, thermal or chemical treatment or by special finishes to meet specific technical or safety requirements, including the following, for example:

- Thermofixing (heat-setting) to give the materials stability of shape.
- Impregnation to make them resistant to moisture, catching fire, clogging with

Table 7: Approximate values for characteristic data of fabric filter media

filter medium	weight g/m²	thickness DIN 53855 mm	air permeability DIN 53887 m³/m²·h
woollen and mixed fabrics	300–400 400–480 480–550 550–650	1.5–2.2 2.2–2.7 2.7–3.3 3.3–4.0	2700–2100 2100–1800 1800–1500 1500– 900
cotton fabrics	100–200 200–300 300–400 400–500	0.5–1.2 1.2–1.7 1.7–2.5 2.5–3.5	1500 1500– 900 900– 600 600– 300
man-made fibre fabrics (polyester, polyacrylonitrile)	180–250 250–330 330–400 400–550	0.3–1.0 0.8–1.5 1.3–2.2 1.9–3.0	2100–1800 1800–1500 1500– 900 900– 600
needle felts (man-made fibres)	250–400 300–500 400–550 500–650	2 –3.5	3600–1800 3000–1800 2400– 900 1200– 360

dust or attack by insects. Such treatment may, however, unfavourably affect the filter properties.

– Admixture of steel fibres and electrically conductive textile fibres, antistatic impregnation or earthing of filter bags, by means of sewn-on metal strands or sewn-in metal wires, to prevent electrostatic charging. Antistatic filter media should, depending on the type of fibre, have a specific electric resistance of 5.0×10^8 to $1.0 \times 10^{18}\,\Omega$cm.
– Finishing of glass fabric with silicone, graphite or PTFE (polytetrafluoroethylene) to improve the mechanical properties and fabric cleaning behaviour (removal of the deposited dust).

The filter media are formed into units shaped like tubular bags or flat envelopes. Dimensional accuracy, careful selection of the material and good execution of the seams are important practical requirements.
Fabric filters used in the cement industry are either **bag filters** (Fig. 8) or screen (or envelope) filters (Fig. 9). Single or multiple filters, used individually or interconnected in series, are used. Casings are circular or rectangular. The term "baghouse" is sometimes applied to large filters containing a number of tubular bags mounted in a usually rectangular casing. As a rule, the dust-laden air is drawn through them by suction. Forced draught is seldom employed.

Table 8: Physical properties of fibres for filter fabrics

type of fibre	natural fibres		man-made fibres					glass fibres
chemical basis	keratin	cellulose	polyamide	polyamide (aromat.)	polyacrylonitrile	polyester	polytetrafluoro ethylene	silicate
name, make	wool	cotton	Nylon Perlon	Nomex	Dralon T	Diolen Trevira	Teflon	glass fabric
density g/cm³	1.32	1.54	1.15	1.38	1.15	1.38	2.1	2.5–2.7
tearing resistance g/den	2.5–5	1–2	4–6	5.5	3.1–3.5	4–7	1.6	7–12
elongation %	25–35	7–10	25–45	20–25	15–30	10–20	18–75	2–5
moisture absorption %	10–15	8–9	4–4.5	2.5–5	1–1,5	0.3–0.4	0	0
water retention capacity (water imbibition value) %	50–70	50–80	10–15	0	8–12	2–5	0	0
melting point °C	>130	>200	250/215	375	>300	250–260	>275	>850
temperature resistance of filter material	80–90	75–85	75–85	190–200	125–135	130–140*)	200–220	250
continuous (max.) °C	(100)	(95)	(95)	(220)	(140)	(160)	(250)	(310)
specific electric resistance Ωcm	$5 \cdot 10^8$	$7 \cdot 10^6$	$4{,}1 \cdot 10^{10}$ – $4.9 \cdot 10^9$	10^{11} – $8 \cdot 10^{13}$	$5 \cdot 10^8$	10^8	10^{18}	10^{15}

*) dry heat

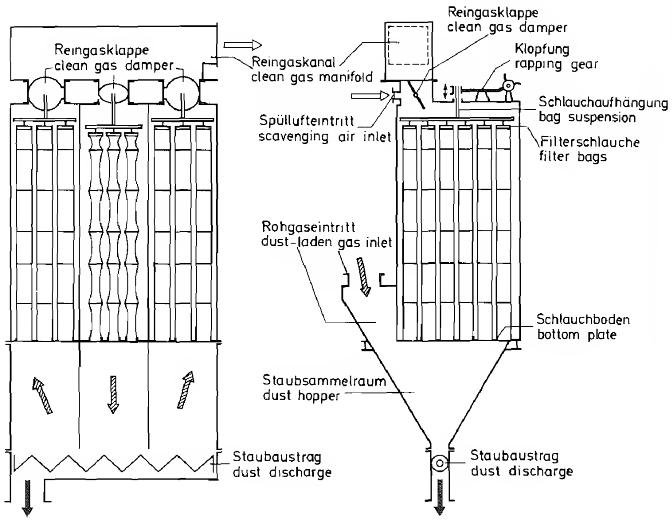
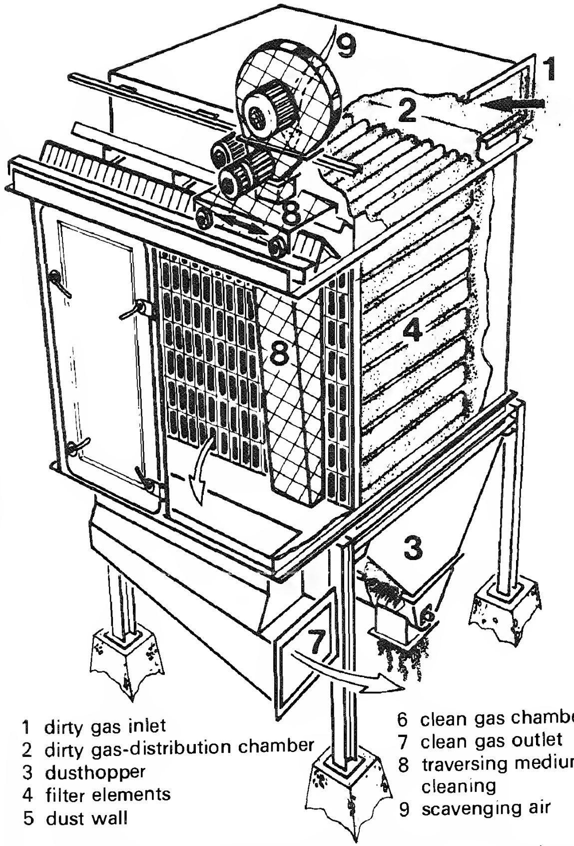


Fig. 8: Bag filter

Cleaning methods for the filter media:

- **Mechanical** cleaning by rapping, shaking or vibration, usually in conjunction with reverse air or gas flow.
- **Pneumatic** cleaning with low-pressure air (reverse flow cleaning of bag filters with cleaned air or gas supplied by a purging air fan), medium-pressure air (supplied in a pulsating flow) and high-pressure air (pulsed compressed air: jet cleaning system).

Filters with **compressed air cleaning** (jet filters: Fig. 10) have come into increasingly widespread use in recent years. These are essentially bag filters in which the bags or tubes are mounted on wire retainers (cages), the dust-laden gas being admitted to the outside of the bags and the clean gas extracted from the inside. For cleaning the bags, a compressed air jet releases a short pulse of air (of 0.1 – 0.2 sec duration) into the top of each bag and entrains a quantity of cleaned air back into the bag, in the reverse direction, i. e., from the inside outwards. These air pulses are applied at regular intervals, adjustable from 1 to 10 minutes. More particularly, the following actions occur in this cleaning method:



1 dirty gas inlet
2 dirty gas-distribution chamber
3 dusthopper
4 filter elements
5 dust wall
6 clean gas chamber
7 clean gas outlet
8 traversing medium pressure impulse cleaning
9 scavenging air

Fig. 9: Screen (or envelope) filter (Heinrich Lühr, Staubtechnik, Stadthagen)

- the normal flow of air through the filter is briefly interrupted and reversed by the pulse of purging air;
- the bag, which in normal operation is collapsed onto its wire cage, is suddenly inflated to its full circular shape (Fig. 11), causing the caked dust to fall off;
- the purging air flows through the filter medium (the wall of the bag) in the reverse direction to the normal flow of dust-laden gas.

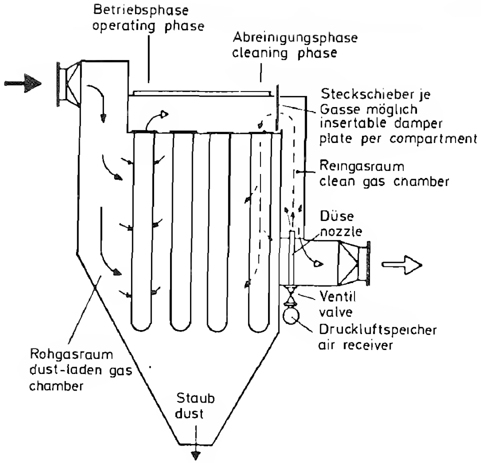


Fig. 10: Bag filter with compressed air cleaning (jet filter)

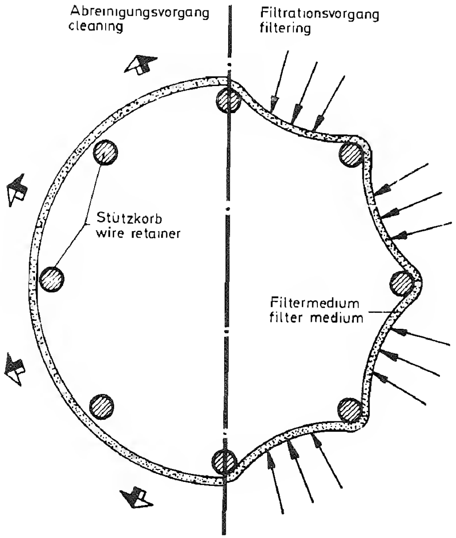


Fig. 11: Filtering and cleaning phases of a jet filter

Besides being effective, compressed air cleaning of the filter medium has the advantage of quickness, so that the proportion of time during which the medium is not available for dust collecting is very small indeed. Besides, it requires less maintenance than other systems. It is especially suitable for use with filter bags made of medium to heavy-grade needle felts with high filtration efficiency.
An important design criterion for filters is the **filter area rating,** by which is understood the volumetric flow rate of the dust-laden gas or air that can be effectively treated per unit area of filter surface, for example: 100 m^3/hour per m^2. This ratio corresponds to the flow velocity of the gas admitted to the filter medium.
Filters subdivided into compartments and designed for mechanical cleaning are cleaned one compartment at a time. The compartment to be cleaned is temporarily disconnected from the flow of dust-laden gas, and therefore the area as well as the area rating must be calculated with due allowance for one compartment less than the total number provided, i.e., the net filter area is the gross (overall) filter area minus the filter area of one compartment.
The purging (or scavenging) air which is admitted to the compartment to be cleaned is subsequently dissipated to the adjacent compartments and must be added to the incoming (dust-laden) gas flow rate for calculating the net filter area rating.
In the case of a filter with compressed air cleaning, however, the net filter area is virtually equal to the gross filter area.
Criteria for selecting a suitable filter medium and rating are:

- temperature and moisture content of the dust-laden gas to be cleaned;
- nature and properties of the dust to be removed,
- type of filter fabric (woven, non-woven);
- form in which the filter medium is used (bag, envelope);
- manner of gas admission (internal, external);
- dust loading of the gas, pre-cleaning and dust distribution in the filter;
- type and from of construction of cleaning system for dislodging deposited dust from the filter medium;
- available amount of space for installing the filter.

The above-mentioned factors more particularly affect:

- the dust content remaining in the cleaned gas;
- the pressure drop through the filter;
- the service life of the filter fabric employed;
- the amount of maintenance required.

Although a low rating and dust loading will require a relatively large filter and therefore higher initial cost, the operating and maintenance costs of such a filter will be correspondingly lower and the filter medium will last longer.
Table 9 gives filter ratings, subdivided according to the method of cleaning, for various filter applications in connection with cement manufacture. Approximate calculation methods are to be found in the relevant literature (see Bergmann, 1976).

Table 9: Filter area ratings

application	type of dust	temp. °C	area rating in $m^3/m^2 \cdot h$ mechanical cleaning	pneumatic cleaning
raw material (comminution, handling, screening, storage)	limestone, marl, clay	<30	100–120	180–200
drying and grinding/drying plants for raw material and coal*)	raw meal, pulverized coal	70–130	60– 80	120–140
handling, storage and feeding of raw meal and pulverized coal*)	raw meal	<50	90–110	150–180
grate coolers for clinker	clinker	<140**)	60– 80	120–140
handling, storage and feeding for clinker*)	clinker	<70	80–100	120–140
cement mill (finish grinding)*) without grinding aid	cement	70–110	80–100	120–140
with grinding aid			60– 80	100–120
handling and storage of cement*)	cement	<70	80–100	120–140
packing plant and bulk despatch loading plant for cement	cement	<50	90–110	150–180

*) With pneumatic handling, high dust content and abrasive dust it is advisable to pre-clean the gas

**) Filter with air cooler installed ahead of it

1.7.1.3 Granular bed filters

Granular-bed filters (see VDI Code 3677) consist of compartments packed with a bed of quartz granules of about 2 to 5 mm size, lying on fine wire mesh in a casing generally of circular shape. In the cement industry such filters are used more particularly for dedusting the exhaust air discharged from grate coolers, as they are resistant to abrasive dust and to high temperatures (up to about 450° C).

The granular bed has to be cleaned at regular intervals, this being done compartment by compartment by means of air in reverse flow, in conjunction with agitation of the granules by rotating agitator arms.

In the granular bed filter with integral cyclone each filter bed is operated with a pre-cleaning cyclone which removes coarse dust particles and also dedusts the air used for cleaning the bed. The combination of the filter bed and the pre-cleaner into a single unit has the disadvantage that, because the compartments are out of action one by one for cleaning the bed, the volume flow rate of dust-laden air admitted to the cyclone tends to be irregular, thus preventing optimum operation (see Berz/Maus, 1977).

In a later development of this type of filter the dust-laden air or gas is pre-cleaned in a separate cyclone (Fig. 12). This air is then distributed to the individual compartments and passed through the granular bed. The finally cleaned air is extracted by a fan and discharged into the atmosphere.

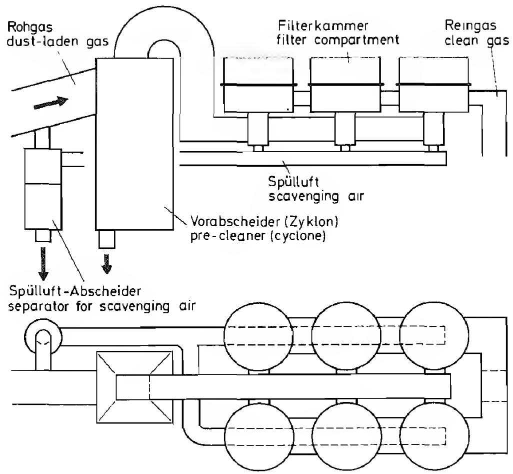


Fig. 12: Diagram illustration the operating principle of the CS granular bed filter

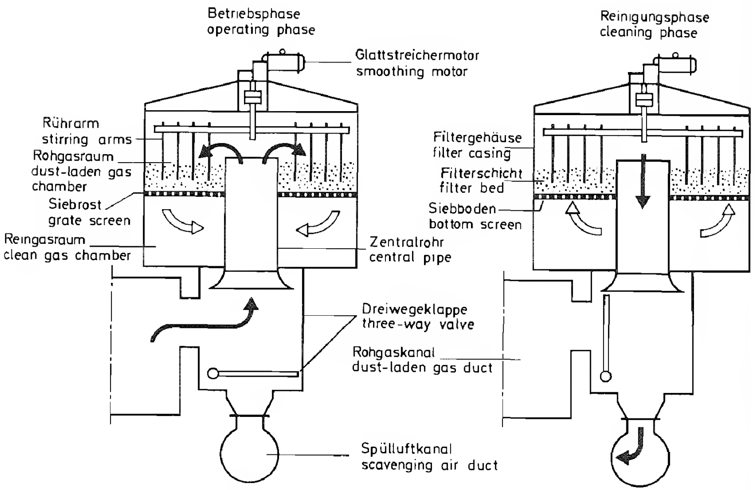


Fig. 13: Operating and cleaning phases of a CS granular bed filter

For the compartmentwise cleaning of the filter the air admission duct is closed by a valve. Next, cleaned air is drawn in reverse flow through the granular bed, which is loosened up at the same time by the rotating agitator. The dust dislodged by the purging (or scavenging) air is precipitated in a special cyclone; the dedusted purging air is then fed back into the incoming air upstream of the pre-cleaner. Fig. 13 shows the operating and the cleaning phase of a granular bed filter of this type.
The pre-cleaning cyclones can be provided with wear-resistant linings. The admission duct for the incoming dust-laden air is so designed that the air flow rate is constant all along its length and that the filter compartments receive equal quantities of air.
One advantage of the granular bed filter is that it can accept high rates of overloading (up to 150%), though this does result in a sharp rise in the overall pressure drop, which under normal operating conditions is 15 – 20 mbar.

1.7.1.4 Electrostatic precipitators

Electrostatic (or electrical) precipitators (see VDI 3678) are used in cement works more particularly for the removal of dust from the exit gases of cement kilns and from the exhaust air discharged by dryers, combined grinding and drying plants, finishing mills and raw mills with water injection.

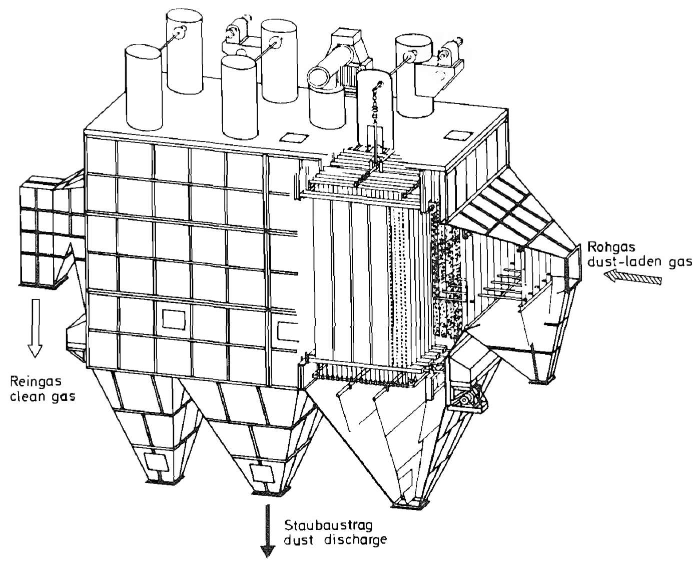


Fig. 14: Horizontal electrostatic precipitator

In the electrostatic precipitator the dust-laden gas is made to flow through a chamber, usually in the horizontal direction, in which it passes through one or more high-voltage electric fields formed by alternate discharge electrodes and plate-type collecting (or receiving) electrodes (Fig. 14). Whereas the latter are earthed, the discharge electrodes are maintained at a high direct (DC) voltage of about 50 to 110 kV, provided by various types of high-tension rectifier equipment.
Under the action of the electric field the dust particles, which have become electrically charged by negative gas ions which are formed at the discharge electrodes and attach themselves to the particles, fly to the collecting electrodes and are deposited there. The dust is dislodged from these electrodes by rapping and thus falls into the receiving hoppers at the base of the precipitator casing.
The electric charges acquired by the dust particles depend substantially on their specific electric resistance. In the lower range of operating temperature the specific resistance increases with rising temperature as a result of diminishing surface conductivity. In the upper temperature range (above about 250° C) the specific resistance decreases, however, as a result of the thermal excitation of conduction electrons, as most dusts have semiconductor properties. Most favourable for precipitation is a specific resistance in the range of about 10^7 to 10^{11} ohm · cm.

The **collection efficiency** of an electrostatic precipitator can be approximately expressed in terms of the well-known Deutsch formula:

$$\eta = 1 - e^{-(w \cdot f)} = 1 - e^{-\left(\frac{t}{a} w\right)} = 1 - e^{-\left(\frac{L \cdot w}{a \cdot v}\right)},$$

where:

η = efficiency
w = particle migration velocity (or drift velocity) in m/sec
f = A/V = specific collecting surface area in sec/m
A = collecting surface area in m^2
$\dot{V}$ = volumetric gas flow rate in m^3/sec
a = distance between discharge electrodes and collecting electrodes: equal to approximately half the collecting electrode spacing (duct width)
t = L/V = retention time of the gas in the electric fields, in sec.
L = sum of the lengths of the collecting electrodes in the gas flow direction, in metres
v = mean gas flow velocity between the electrodes, in m/sec.

The particle **migration velocity** (drift velocity) is a reference quantity comprising all the influences on the dust collecting performance, e. g., specific resistance of the dust, electric field intensity, corona discharge current, dust particle size, viscosity of the gas, design features of the electrodes, design of the precipitator, etc. The migration velocity must therefore be estimated on the basis of experience gained with comparable other installations by expert engineers.

Typical gas and migration velocities in electrostatic precipitators as used in the cement industry are listed in Table 10. These figures are based on a dust content of less than about 0.1 g/Nm^3 (dry) in the cleaned gas.

Table 10: Gas velocity and particle migration velocity in electrostatic precipitators for cement works

type of plant	mean gas velocity m/s	migration velocity m/s
wet-process rotary kiln	0.9–1.3	0.1 –0.13
rotary kiln with grate preheater	0.8–1.2	0.08–0.12
rotary kiln with cyclone preheater and conditioning tower	0.8–1.2	0.08–0.12
rotary kiln with cyclone preheater and exit gas utilization	0.6–1.0	0.06–0.10
shart kiln	0.6–1.0	0.06–0.10
grate cooler	0.5–0.8	0.04–0.08
dryer, grinding/drying plant	0.6–1.2	0.06–0.12
tube mill with water injection	0.6–1.0	0.06–0.10

The casing or housing of the precipitator is constructed of steel plate or reinforced concrete and is, as a rule, provided with 50–150 mm thick thermal insulation. Uniform gas distribution over the full inlet cross-section is ensured by means of suitable internal fittings (e. g., perforated baffles). The dust receiving hoppers are provided with devices for the prevention of flow phenomena within them.

Depending on the desired discharge behaviour, the **discharge electrodes** are given various shapes. straight or helical round wire, specially profiled wire of square or star-shaped cross-section, wire or strip with saw-tooth pattern or spiky projections, or tubes with pointed projections. These electrodes are mounted in rigid supporting frames or are freely suspended; in the latter case the lower end of each discharge electrode is fastened to a tautening weight.

The **collecting electrodes** (receiving electrodes) generally consist of metal plates, which may be plain or be specially profiled to give maximum surface area, reduce dust re-entrainment and permit effective rapping.

The distances between the discharge and collecting electrodes are usually in the range of 125 to 150 mm, exceptionally much larger (250 mm and more). The material most commonly used for the electrodes is steel. In certain cases, however, various special materials are used, e. g., aluminium alloys or highly corrosion-resistant steel.

The duration and intensity of **rapping** the electrodes will be adapted to the properties and quantities of the dust concerned. Individual rapping for the respective electric fields is preferable.

The precipitator is **energized** by a system comprising a step-up transformer (to provide the required high voltage from the normal AC power supply) and a rectifier, which are accommodated in one cubicle either in the high-voltage equipment room or on top of the precipitator itself. Various types of rectifier are available, but the type most extensively used for the present purpose is the static rectifier using silicon diodes. A switch cubicle is also provided. The high-voltage energizing system is equipped with transductor or thyristor-controlled voltage regulation and operated a little below the flashover voltage (Fig. 15). This close voltage control is necessary because the collection efficiency is governed to a great extent by the voltage applied. The criterion for the flashover limit is the flashover itself or the frequency of flashover discharges. With single-phase AC supply the arcing is more readily extinguished than with three-phase.

The number of energizing units to be provided will depend on the need for achieving optimum dust collecting performance and the availability of adequate standby capacity.

The insulators at the high-voltage entries may become faulty as a result of dirt deposits forming on them, resulting in leakage currents. To counteract these troubles, the insulators may be heated, screened and/or swept with gas or air.

The dust receiving hoppers may be of the tapered or the trough type and are equipped with discharge devices such as rotary gates, screw conveyors or continuous-flow conveyors with dust locks. If necessary, the dust hoppers and discharge devices can also be thermally insulated and moreover be electrically heated

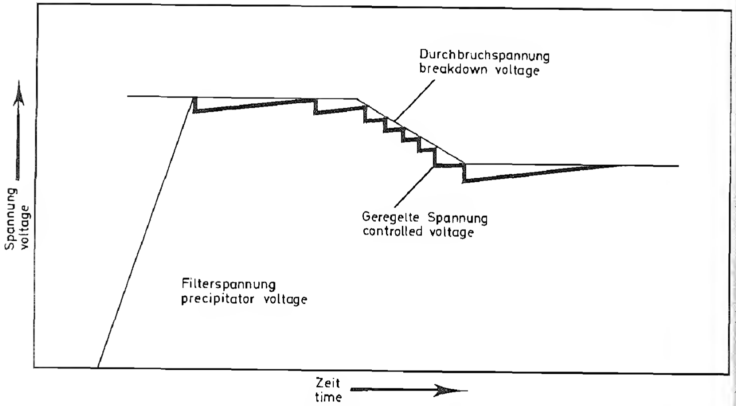


Fig. 15: Example of automatically controlled precipitator voltage

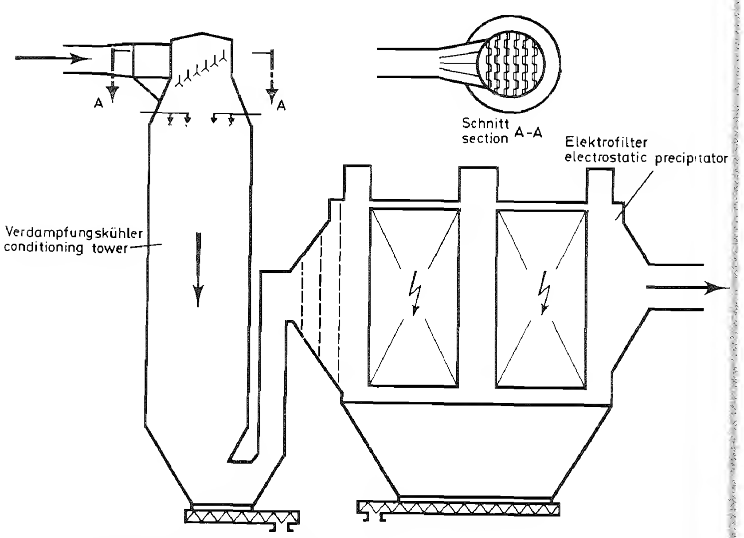


Fig. 16: Electrostatic precipitator with conditioning tower

As a precaution against explosion or fire hazard the precipitator casing may be of explosion-resistant construction, with venting panels on its roof and sides.

A **conditioning unit** (water spray tower) installed upstream of the precipitator can serve to lower the temperature of the incoming gas to 130°–150° C and also raise the dew-point of the gas, so as to enable better dust precipitation to be achieved. At the same time, the gas entering the conditioning unit should still have a sufficiently high entry temperature, uniform distribution and sufficiently long residence time to enable all the sprayed water to be completely evaporated (Fig. 16).

The conditioning of the gas requires about 0.5 g of water per Nm^3 and per degree. As a rule, the water is injected through return-flow nozzles arranged in a ring at the top of the tower-type unit, so that the droplets travel in parallel flow with the downward stream of gas. The rate of water discharge from these nozzles (normally within a 1:10 range) is controlled with reference to the temperature of the gas at the outlet of the conditioning unit, but in some cases additionally with reference to the volumetric flow rate of the gas. Despite good servicing and control, moisture may cause dust agglomerations (lumps) in the conditioning unit. In order to prevent possible choking, the dust discharge devices should therefore be designed to sufficiently large cross-sectional dimensions.

The **pressure drop** through electrostatic precipitators is between about 1 and 3 mbar, depending on the type of internal fittings. The conditioning unit additionally has a pressure drop of about 5 mbar. For fairly high dust loadings in the gas (above about 100 g/Nm^3) a mechanical separator is generally installed as a pre-cleaner upstream of the electrostatic precipitator or incorporated within it (ahead of the first electric field).

Electrostatic precipitators of the vertical-flow type may alternatively be used for dedusting the exhaust air from **coal grinding/drying plants.** In the event of deflagration or explosion, the pressure wave generated by such phenomena can be effectively dissipated upwards. Besides, in precipitators of this type, the risk of potentially dangerous dust deposits forming in the interior is small. However, in comparison with horizontal-flow precipitators, the gas velocity will as a rule have to be lower. Another drawback of the vertical-flow precipitator is that it is not possible to provide separate electric fields with separate rapping arrangements.

1.7 1.5 Operation and maintenance of separators and precipitators

Every dust collection plant with its ancillary equipment must be properly operated and serviced so as to enable it to perform its duties at all times. The manufacturers of such plant should accordingly, when erecting and commissioning it, supply the user with the following documents:

- description of the plant with drawings showing the construction, arrangement and functioning of its parts and of the measuring and control instrumentation,
- operating instructions for starting and stopping, normal running, and the detection and elimination of faults;
- instructions for the servicing and lubrication of the various pats of the plant;

– maintenance and repair manual with a list of instructions for the fitting of spares and renewable parts.

General guidelines for the operation and maintenance of such plant are given in VDI Code 2264. Special instructions for individual types are contained in the following VDI Codes:

VDI 3676 Inertia-force separators
VDI 3677 Filters
VDI 3678 Electrostatic precipitators.

Instructions for the erection and operation of electrostatic precipitators are moreover given in VDE 0146 and 0105 (Part 8). With regard to the prevention of dust explosions and fire outbreaks and to pressure relief (venting) arrangements for dust explosions VDI 2263 and 3673 should be consulted.

1.7.2 Combating in-plant dust sources

Major sources of dust arising within the cement works are the silos, the indoor or outdoor stockpiles for bulk materials containing fine particles, the despatch loading installations of clinker and cement, the unloading installations for bulk materials containing fine particles, and the roads on the cement works site.
Ways and means of preventing dust formation at such sources and of abating the resulting environmental nuisance are suggested in VDI 2094. Other guidelines are given in the "Technische Anleitung zur Reinhaltung der Luft" ("Technical directives for clean air") (1974).

2 Noise control

2.1 Technical principles

2.1 1 Definitions (see Schmidt, 1976)

Sound. Mechanical vibrations in an elastic medium.
Airborne sound: Sound which is propagated in the form of sound waves in air, these being produced by variations in air pressure.
Velocity of sound in air at 20°C: 343 m/s (metres per second).
Solid-borne sound (structure-borne sound, impact sound). Sound which is propagated in a solid medium, or at the surface thereof, with frequencies of more than about 15 Hz, produced by vibrations of the medium. In the case of lower frequencies these phenomena are referred to as vibrations or oscillations.
Frequency (f): The The number of cycles (complete vibrations) per second. The hearing range of the human ear is between about 16 Hz and 18000 Hz (in young people) and 12000 Hz (in older people). A range of about 350 to 1500 Hz is sometimes called the medium frequency range, with low frequency and high frequency below and above these limits respectively The unit of frequency is the Hertz (Hz), equivalent to one cycle per second.

Octave: The interval between two frequencies which are in the ratio of two to one, i.e., two frequencies f_1 and f_2 such that $f_2 = 2\ f_1$.
Third: One-third of an octave, i.e., two frequencies f_1 and f_2 such that $f_2 = \sqrt[3]{2}\ f_1 = 1.26\ f_1$.
Tone: Sound with sinusoidal pattern of its vibration amplitude and with a frequency within the range of audibility (hearing range).
Timbre: Denotes the quality of sound composed of a number of tones whose frequencies are whole multiples of a fundamental frequency.
Noise: Sound composed of many tones of random frequency.
Noise· Any kind of sound that is objectionable or harmful to human beings.
Sound pressure (p): The additional pressure (superimposed upon the normal atmospheric pressure) due to the compression of the air associated with the propagation of sound. At the threshold of human audibility at 1000 Hz the sound pressure is:
$p_o = 2 \times 10^{-5}\ \mathrm{N/m^2}$.
Sound pressure level (L_p) or simply "sound level": 20 times the logarithm (to the base 10) of the ratio of the pressure p_n of the sound to the reference pressure p_o:

$$L_p = 20 \log \frac{p_n}{p_o}\ \text{(dB)}.$$

Decibel (dB): Unit of sound level using a logarithmic scale.
Sound power (W). The acoustic power emitted by a source of sound or noise. It cannot be directly measured, but is calculated from the sound pressure p (in mbar) and the measuring surface area F (in m²): $W = 0.025\ p^2 \cdot F$ (in mW).
Sound intensity (J) A measure of the sound power per unit area perpendicular to the direction of propagation; for a free sound field:

$$J = p^2/\varrho\ c = 2.45 \times 10^{-3}\ p^2 = w\ c\ \text{(in W/m}^2\text{)}.$$

where:

p = sound pressure in $\mathrm{N/m^2}$
$\varrho\ c$ = wave impedance = 410 $\mathrm{Ns/m^3}$ under normal conditions in air
w = sound energy density in $\mathrm{Ws/m^3}$
c = sound velocity in m/s.

The sound intensity is proportional to the square of the sound pressure $J/J_o = p^2/p_o$.
Sound power level (L_W): 10 times the logarithm (to the base 10) of the ratio of the sound power W radiated by a source to a reference power W_o.

$$L_W = 10 \log \frac{W}{W_o}\ \text{(dB)},$$

where $W_o = 10^{-12}$ watts.
The sound power level is not a directly measurable quantity. It is calculated from the sound pressure level L_p and the measuring surface ratio L_s of the sound-emitting source:

$$L_w = L_p + L_s\ \text{(dB)}$$

Sound intensity level (L_J): $L_J = 10 \log \frac{J}{J_o}$ (dB),

where: $J_o = 10^{-12} W/m^2$ (watts per m^2).

Measuring surface ratio (L_s): 10 times the logarithm (to the base 10) of the ratio of the measuring surface S (in m^2) enveloping a sound-emitting source to the reference surface area S_o ($= 1\ m^2$):

$$L_s = 10 \log \frac{S}{S_o} \text{ (dB)}.$$

Equivalent continuous sound level (L_m): Mean value of measured sound pressure levels (according to DIN 45641), corresponding to the acoustic effect of fluctuating noise over the period of time of the measurements.

Effective level: Mean value of the maximum sound pressure levels over a certain period of time, as defined in the noise control code "TA Lärm" (1968).

Assessment level: Effective level relating to particular times (day, night): according to "TA Lärm", it is subject to an addition (of up to 5 dB) to allow for individual tones and to a reduction (3 dB) in respect of uncertainty in the measurements.

Impulse noise: Noise comprising rapid increases (of at least 5 to 10 dB) of short duration in the sound level. The impulse sound level L_J can be measured with appropriate equipment. If an ordinary sound level meter (not an impulse sound level meter) is used for determining the equivalent continuous level L_m, the measured value should be increased by 6 dB.

Individual tone. Any objectionable tone or note distinctly emerging from the noise emitted by a source.

Sound insulation: Prevention or reduction of the transmission of sound through a partition (wall, ceiling, etc.). Most of the sound is reflected from the partition.

Sound reduction index (transmission loss): A logarithmic criterion for the sound insulation of a partition·

$$R = 10 \log \frac{W_1}{W_2} \text{ (dB)}.$$

where W_1 is the sound power incident on the partition, and W_2 the sound power transmitted through it.

Approximately (Schmidt, 1976). $R = -25 + 18 \log g + 12 \log f$ where g is the weight per unit area of the partition (g/m^2) and f is the frequency (Hz).

A mean sound reduction index R_m is often indicated which (for sound incidence from all directions) is obtained as the arithmetical mean in the practical range of 100 to 3200 Hz adopted in architectural acoustics. As a rule, it corresponds to insulation at a frequency of 500 Hz. According to DIN 52210, Sheet 4, the adjusted value R_w should preferably by used as a single-figure rating to indicate the sound reduction. For "single-leaf" partitions, approximately: $R_w = R_m - 2$.

Sound absorption: The attenuation of sound waves on impingement upon sound-absorbing interfaces, due to friction and conversion into heat.

Sound absorption coefficient: A measure of the absorption of sound by structural components and rooms.

Insertion loss (D_e): The reduction of sound level by the introduction of a control device, e.g., a sound attenuator. Measured by the substitution method of testing, i.e., sound measurement performed with and without the control device inserted.

Weighting: The application of standard specified corrections or adjustments to measured values in order to take account of the human ear's differing responses to frequencies, intensities and duration of acoustic stimuli. Hence the purpose of weighting networks on measuring instruments is to make the readings correspond as closely as possible to the perceived noise level.

Weighting for duration: The adjustment is made by speed settings of the measuring instruments: "slow", "fast", "impulse".

Frequency weighting: Weighting for loudness and type of sound by means of various weighting networks fitted in the sound level meter; mainly these are the A, B, C and D weightings (weighting curves are given in DIN 45633).

For the characterization of noise the A-weighted sound level (L_A) is now almost exclusively employed (values in A-weighted decibels: dBA), i.e., the A-weighting is now specified for the rating of sounds irrespective of loudness level and is no longer restricted to low-level sounds.

Loudness is a subjective characteristic of a sound. The unit of loudness level is the phon. The loudness level (in phons) of a sound is numerically equal to the pressure level (in decibels) of a 1000 Hz pure tone which is judged by the average observer to be equally loud.

The frequency of 1000 Hz is thus the reference for all loudness measurements, and all contours of equal loudness (in phons) have the same numerical value as the sound pressure level at 1000 Hz. For example, a 50 dB tone at 1000 Hz has the same loudness level (50 phons) as a 73 dB tone at 50 Hz or a 42 dB tone at 4000 Hz.

2.1.2 Addition of sound levels

Since the sound pressure level (= sound level or noise level) is a logarithmic ratio, the resulting sound level from two or more sources located close to one another cannot be obtained by straight addition. Thus, the resulting ("total") sound level from n sources of equal intensity J_1 will be:

$$L = 10 \log n \frac{J_1}{J_0} = 10 \log \frac{J_1}{J_0} + 10 \log n = L_i + 10 \log n \text{ (dB)}.$$

For two sources of sound (n = 2) we thus have:

$L = L_i + 10 \log 2 = L_i + 10 \times 0.3 = L_i + 3$ (dB).

Therefore: new sound level = old sound level + 3 dB.

Doubling the number of (equal) sources increases the sound or noise level by 3 dB.
For n equal sources, each having the same sound pressure level L_i, distributed within a given space the mean level L in that space is approximately:

$L = L_i + 5 \log n.$

The total sound level L_n of n equal sound sources located at equal distances from the point of measurement is obtained as the sound level L_o of one source plus a value ΔL:

$L_n = L_o + \Delta L.$

Table 11 gives values for ΔL.

Table 11: Additional term ΔL depending on the number n of sound sources

n	2	3	4	6	8	10
ΔL	3	3.8	6	7.8	9	10

Determining the total sound level of several sources not of equal intensity, or the total sound level from the octave or third levels, can be done with the aid of the intensity ratio $g = J_1/J_2$ (rating factor), as listed in Table 12, which gives values of g for different values of ΔL, i.e., the difference in level between the individual sound levels L_i and the reference level L_o. As a rule the lowest of the sound levels under consideration is taken as the reference level. Thus:

$$L_{total} = 10 \log \sum_{i=1}^{n} 10^{0.1 L_i}\ (\text{dB}) = 10 \log \sum_{i=1}^{n} g_i\ (\text{dB})\,.$$

Worked example:

Reference level $L_o = 80$ dBA

Individual levels L_i	$\Delta L = L_i - L_o$	g from Table 12
85 dBA	5 dB	3.2
80 dBA	0 dB	1.0
87 dBA	7 dB	5.0
90 dBA	10 dB	10.0
92 dBA	12 dB	16.0
84 dBA	4 dB	2.5

$\Sigma g = 37.7$

corresponding value of ΔL from Table 12 = 16 dB (approx.).

Hence: $L_{total} = L_o + \Delta L = 80 + 16 = 96$ dBA.

Table 12: Relationship between sound level difference ΔL and intensity ratio (rating factor g)

$\Delta L_{(dB)}$	g	$\Delta L_{(dB)}$	g	$\Delta L_{(dB)}$	g	$\Delta L_{(dB)}$	g
0	1.0	10	10	20	100	30	1000
1	1.3	11	13	21	130	31	1300
2	1.6	12	16	22	160	32	1600
3	2.0	13	20	23	200	33	2000
4	2.5	14	25	24	250	34	2500
5	3.2	15	32	25	320	35	3200
6	4.0	16	40	26	400	36	4000
7	5.0	17	50	27	500	37	5000
8	6.3	18	63	28	630	38	6300
9	8.0	19	80	29	800	39	8000

If the number of sources is small, the addition procedure for arriving at a total sound level can be simplified by using a nomogram (Fig. 17). The higher sound level is increased by an amount depending on the difference in level. If there are more than two levels, the procedure is done step by step. It is advisable to perform the addition with values of one decimal place and round the final result to a whole number of dB.

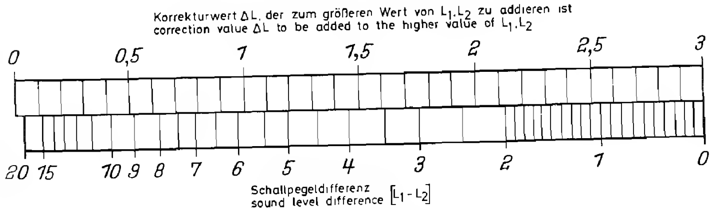


Fig. 17: Nomogram for the logarithmic addition of sound levels

The significance of the difference in level ΔL between two noise levels L_1 and L_2 is shown by the following figures:

$\Delta L = L_2 - L_1$		Significance
1 dBA	=	Just perceptible difference in loudness between the two noises
3 dBA	=	Halving or doubling of the sound level (two sound sources of equal intensity)
6 dBA	=	Four sources of equal intensity
7 dBA	=	Five sources of equal intensity
10 dBA	=	Ten sources of equal intensity: halving or doubling of the (subjective) loudness perception

2.1.3 Averaging of sound pressure levels

If a number of measured values of equivalent accuracy are available, the determination of the average or mean can be done arithmetically, so long as the sound levels do not differ by more than 10 dB. If the differences are larger, quadratic (i.e., energy-related) averaging will have to be applied. The rules of sound level addition can be adopted for the purpose; the average is given by:

$$\tau = 10 \log \frac{1}{n} \sum_{i=1}^{n} 10^{0.1 \cdot L_i} = L_{total} - 10 \log n \ \text{(dB)}$$

First, all the measured sound levels L_i should be added with the aid of the factors g (Table 12) to give the total level L_{total}. From this result must then be subtracted the value $L_n = 10 \log n$ (in dB), where n is the number of measured values.

Worked example:

Total sound level:	96 dBA
Number of measured values:	n = 6
Average sound level:	$\tau = 96 - 10 \log 6$
	$= 96 - 7.8 =$ **88 dBA** (rounded).

For the assessment of greatly **fluctuating** sound levels, e.g., traffic noise, the equivalent continuous sound level L_m over the measurement period will have to be determined. It is based on the principle that there exists a continuous steady noise level which is "equivalent" in its objectionable effect or nuisance effect to the fluctuating noise and whose level is equal to the acoustic energy mean of the fluctuating noise:

$$L_m = \frac{g}{0.3} \log \left(\frac{1}{T} \sum_{i=1}^{n} t_i \cdot 10^{0.3\,L_i/q} \right) \text{ in (dBA)}$$

where:

q = halving parameter, indicating that increase in sound level which, if the duration of the sound is halved, is perceived as equally loud or objectionable; according to VDI 2058 and "TA Lärm" (1968): q = 3 dB

$$T = \sum_{i=1}^{n} \cdot t_i$$

T = total measurement time (reference period of time)
t = interval times.

For the energy-related averaging procedure which is predominantly applied, using q = 3, the above expression becomes:

$$L_m = 10 \log \left(\frac{1}{T} \sum_{i=1}^{n} t_i \cdot 10^{0.1 \cdot L_i} \right) \text{(dBA)}$$

Averaging can be done with the aid of the rating factors g if the measured values have been obtained by successive readings on a hand-held sound level meter:

$$L_m = L_o + 10 \log \bar{g}, \text{ where } \bar{g} = \frac{1}{T} \sum_{i=1}^{n} g_i \cdot t_i \ \text{(dBA)}$$

Quite often, integrating measuring instruments are used in combination with sound level meters, or the necessary averaging calculations may be performed by an electronic computer.
In the random test procedure the sound level is measured at certain points of time – e.g., at intervals of 0.1, 0.3, 1 or 5 sec, or even longer intervals appropriate to the circumstances.
In the "maximum level" averaging procedure described in "TA Lärm" (1968) the maximum sound level which occurs in each time interval of duration t (as a rule: t = 5 sec; but for purposes of audiometry in industry, more particularly for protecting people at work against noise: t = 3 sec) is used for determining the equivalent continuous level. Worked examples illustrating this method are given in "TA Lärm".
If a number of time intervals t_i with respective sound levels L_i are considered, the equivalent continuous sound level L_m for the total measurement time T (e.g., from 6 a.m. to 10 p.m. for daytime measurements, or from 10 p.m. to 6 a.m. for measurements at night) is calculated from:

$$L_m = 10 \log \left(\sum_{i=1}^{k} \frac{T_i}{T} \cdot 10^{0.1\,L_i} \right) \text{(dBA).}$$

In the exceptional case where a plant or piece of machinery causes a sound level L over a period t and the equivalent continuous sound level over a period T has to be calculated from this, the formula to use is:

$$L'_m = 10 \log \left(\frac{t}{T} \cdot 10^{0.1 \cdot L}\right) = L - 10 \log \frac{T}{t} \text{ (dBA).}$$

Worked example:

The mean sound level emitted by a building accommodating a clinker grinding plant is (in this example) 80 dBA. The plant operates only during four night hours:

t = 4 hours
T = 8 hours (night-time assessment period envisaged in the "TA Lärm" regulations)
L = 80 dBA.

The time-corrected equivalent continuous sound level is then:

$$L'_m = 80 - 10 \log \frac{8}{4} = 80 - 3 = \mathbf{77\ (dBA).}$$

The above-mentioned relationships can also be used in a case where a noise source which could not be representatively measured during the time of its occurrence has to be additionally allowed for in the calculation of the equivalent continuous level, for example: traffic noise on the site of a cement works to be added to the noise emitted by the works itself.
The equivalent continuous sound level can comprise a measurement period of any length. In order to assess the permissibility of a noise, i. e., whether or not it can be tolerated, it must be determined over the entire "assessment period". For noise affecting adjacent residents this period is usually specified at 16 hours by day and at 8 hours by night ("TA Lärm") or at 1 hour (VDI 2058, Sheet 1). For purposes of industrial audiometry (exposure of persons to noise in their place of work) the period is generally 8 hours (VDI 2058, Sheet 2).
A relatively constant noise level need not be measured during the whole of this assessment period, but only for short lengths of time. The timing and duration of these measurements are to be so chosen that the result is **representative** of the whole period they are intended to cover.

2.1.4 Extraneous noise, background noise

Extraneous noise denotes the noise emitted by sources other than those which have to be measured. The extraneous noise level L_F is determined while the source or sources to be measured (in general: machinery or some kind) are stopped. If the

total noise level L_G is less than 3 dB above L_F, the measurement for determining the level of the noise source to be measured cannot be evaluated.
If the difference between L_G and L_F is at least 3 dB, i.e., $\Delta L = L_g - L_F \geq 3$ dB, the following correction levels should be subtracted:

difference ΔL:	3	4 to 5	6 to 8	dB
correction level:	3	2	1	dB.

If possible, sound immission measurements – i.e., measurements for ascertaining the degree of nuisance experienced at particular points – should be performed at times when extraneous noise levels are low (e.g., at night).
The lowest extraneous noise level during the measurement time is called the background level (VDI 2058, Sheet 1). The background noise level L_{90} is that level which is exceeded during 90% of the measurement time; L_{95} is similarly defined. Quite often the background noise level in the neighbourhood of a cement works corresponds to the uniform level due to the noise emission from the works itself.

2.1.5 Sound propagation; obstacles

In a free field, sound emitted by a point source travels uniformly in all directions, i. e., the wavefronts are spherical, while the sound level decreases by 6 dB per doubling of the distance from the source.
In general, with an increase of distance from r_1 to r_2 (in m) the sound level decreases from L_1 to L_2, as follows:

$$L_2 = L_1 + a \frac{10}{3} \log \frac{r_1}{r_2} \text{ (dB)}$$

where a = 6 dB for spherical wavefronts spreading uniformly in all directions and a = 3 dB for cylindrical wavefronts spreading in planes perpendicular to a line source of infinite length. If the line source is of finite length l, the wavefronts in the immediate vicinity of it are quasi-cylindrical, but approximating more and more closely to spherical wavefronts with increasing distance, until at a distance $r = l/\pi$ they can be regarded as virtually spherical.
In hall-type factory buildings the value of the factor a is only about 2 to 4 dB. For an average absorption coefficient α in the building: $a = 20\,\alpha$.
If a noise source is of considerable size (e.g., a large machine or the wall of a building), the sound radiating surface area S will have to be taken into account in calculating the decrease in sound level with increasing distance. The sound level L at a distance r (in m) from the source, without considering directivity, is then:

$$L_r = L_s - a \frac{5}{3} \log \frac{2 \cdot \pi \cdot r^2}{S} \text{ (dB)}$$

where L_s denotes the sound level directly in front of the surface S (area in m^2).

The approximate calculation of sound propagation from plane sources (industrial areas, factory sites) is explained in DIN 18005, Part 1; see also Funke, 1978.
The detailed calculation of sound propagation from machinery, plant and buildings can be carried out in accordance with VDI 2571 and 2714.
Noise attenuation can be achieved with the aid of **acoustic screens** such as walls, buildings, earth embankments, hillocks, etc.
The effectiveness of a screen or barrier wall depends on its effective screening height H, its distance from the noise source and from the point of immission, and the frequency composition of the noise. The screening effect is greater according as the difference in length between the direct line connecting the sound source to the point of immission and the detour path around the obstacle is greater. The attenuation achieved by screening can be calculated in accordance with DIN 18005, Part 1, and VDI 2571 and 2714.
Attenuation values of 15 to 25 dB due to screening can be obtained in the immediate vicinity of a noise source. At greater distances, above about 200 m, the practically attainable noise attenuation by screening is approximately 10 dB.
In regions with "open" building development (residential areas, factory sites) an additional noise level reduction of 5 to 10 dB per 100 m of such development traversed by the sound can be assumed (DIN 18005, Part 1). For the calculation of sound attenuation in the downwind direction under conditions of temperature inversion, however, the presence of buildings is not taken into account except when they occur in a built-up area exceeding an arc of 5 km radius.
As a general rule, the low frequencies of a noise are much less effectively screened by acoustic obstacles than are the medium and high frequencies.
Dense forest or high and dense shrubbery of considerable depth can reduce the sound level by scattering and absorption. The attenuation due to such vegetation can be calculated from:

$$\Delta L_w = 0.01 \sqrt[3]{f} \text{ (dB per m)},$$

where the frequency f is in Hz. The attenuation is here, too, greatly restricted in the downwind direction if there is appreciable wind and temperature inversion (arc of 5 km radius).
Sparsely distributed trees, bushes, hedgerows, etc. have very little sound-attenuating effect.

2.2 Sources of noise in cement works

Table 13 lists the principle sources of noise in cement works (VDI 2714, Sheet 1; Techn. Merkblatt, Bundesverband der Deutschen Kalkindustrie, 1975) with the sound pressure levels associated with them. The actual values depend more particularly on the nature, size and capacity of the machines and on the their running speed, pressure rise developed, feed material and other factors significantly affecting noise emission. The figures given in the table comprise the actual machinery noise as well as any working noise emitted in connection with them. Frequency spectra for individual noise sources are given in the literature (see Funke, 1969, 1973, 1977 and 1978).

Table 13: Pressure levels of sound sources in cement works

sound source	measuring distance m	sound pressure level dBA
percussion drilling machines (pneumatic)	1	100–115
rotary drilling machines (pneumatic)	1	90–100
rope-operated excavators (diesel)	7	85– 95
rope-operated excavators (electric)	7	75– 90
hydraulic excavators (diesel)	7	90–100
wheel loaders, crawler loaders	7	85–105
bulldozers, rippers	7	85–105
heavy trucks	7	85– 95
hammer crushers	1	100–110
impact crushers, impact mills	1	85–100
tube mills	1	100–115
roller mills	1	90–105
mill drives	1	90–105
screening machines	1	95–115
planetary coolers	4	80–105
rotary coolers	1	80– 90
kiln firing systems (kiln hood)	1	90–100
rotary kiln drives	1	85– 95
gas pressure reducing plants	1	95–100
natural gas pipelines	1	85– 95
fans, intake or discharge	1	95–125
fans, casings	1	75–105
rotary piston blowers	1	100–120
reciprocating piston compressors	1	95–110
rotary compressors, screw compressors	1	100–120
water pumps	1	85– 95
drives of conveyors	1	80– 95
bucket elevators (sheet-steel casings)	1	85– 95
rubber-belt conveyors	1	65– 80
vibratory conveyors	1	85–100
chutes for materials	1	85–110
belt conveyor transfer points	1	90–105
packing machines	1	75– 85
automatic palletizers	1	80– 90
bulk carrier vehicles	7	80– 90
lift trucks (diesel)	7	80– 90
lift trucks (electric, gas)	7	60– 75

2.3 Noise abatement

2.3.1 Basic measures

Primary noise control comprises measures for dealing with noise at the source: suppression or modification of causes and reduction of sound emission and transmission Where such measures are inadequate for the purpose, they must be supplemented by **secondary noise control**, which strives to prevent or reduce sound propagation.
The following are some basic points and considerations relating to noise control:

- Choose machinery and methods that emit the least possible noise.
- Apply appropriate design measures to ensure most favourable location of noise sources in relation to immission points.
- Apply the principle of concentration of noise sources.
- If possible, begin by reducing the noise source that determines the noise level at the relevant immission point.
- Reduce impulse noises, individual tones and high-frequency noise components.
- Apply secondary noise control measures (e.g., acoustic screening) as close to the source of noise as possible.
- Combine noise control measures with measures and precautions to promote safety, clean air, thermal insulation, vibration insulation, etc.
- Avoid carrying out noisy industrial or other activities at night.

At the same time, some obvious plant operational requirements will have to be satisfied Thus noise control measures must not·

- present a hazard (explosion, fire, accident);
- impede plant operation, accessibility and maintenance,
- obstruct heat dissipation and gas discharge.

Furthermore, in connection with noise control measures it is necessary to take account of the additional space requirements involved and also of such matters as strength, durability (e.g., corrosion protection), wear and possible choking of sound attenuating devices, which would thus become ineffective.
The various available measures for reducing noise generation and transmission and for restricting the noise radiated from machinery and other installations are reviewed in VDI 2570. The arrangements which are of most importance in cement manufacture, and have been duly put to the test in actual practice, will be briefly described below.

2.3.2 Taking account of noise control at the plant design stage

In addition to the points indicated in Section 2.3.1, the following aspects should be given due consideration in connection with cement works planning:

- The significance of the cement works as a possible source of noise nuisance in connection with the planning of adjacent new residential areas and with regional or local development plans.

- Keeping noisy machinery or other installations as far away from adjacent residents as possible.
- Screening of noise sources by the interposition of buildings (e.g., silo installations forming a closed continuous barrier).
- Installing noisy machines in closed buildings.
- Avoiding any noise sources mounted at considerable height above ground level.
- Restricting the number of doors, windows and ventilation openings in the buildings to the essential minimum; also, the doors, etc. should if possible be located on the side facing away from the adjacent residents, taking due account of sound reflection from the walls of any other buildings at the rear.
- For the environmental comfort of plant operating personnel, the control rooms should adequately sound-insulated.

2.3.3 Measures and precautions for machinery

Reduction of structure-borne noise propagation: Separation of machinery from the foundation of the building by means of joints filled with vibration-isolating material and sealed against ingress of dirt. Elastic elements such as rubber pads, rubber seals, sleeves, plastic elements. Avoidance of structural sound bridges along which noise can be transmitted; resilient mounting of sheet-metal parts. Spring elements for reducing structure-borne sound transmission in the low range of frequencies. For other vibration-isolating elements see VDI 2062, Sheet 2. Emission of structure-borne noise from surfaces can be suppressed by making these of flexible construction. Using materials with high internal damping. Applying anti-drumming or sound-absorbing facing layers.

Low-noise design, precision manufacture: High accuracy of machining and surface finish for machinery parts revolving against one another, to ensure quiet running. Rotating masses should be properly balanced. Power transmission to be effected through acoustically favourable devices such as flexible couplings or fluid couplings. Use of helical gears with low moduli. Choice of gear combinations in which one of any pair of gear wheels consists of material possessing high internal damping. Quiet-running bearings should be used (e.g., plain bearings are generally quieter than antifriction bearings), with minimal play. Rattling parts should be fixed. Cast-iron housings less noisy than welded ones. Stiffening ribs on sheet-metal panels. Good alignment of gear stages. No holes in flywheels and belt pulleys (such holes are liable to cause high-pitched whine).

Electric motors: Motors with low speeds are preferable. Big motors to be water-cooled rather than air-cooled. Direct drive by four-pole or variable-speed motors. Choose motor types with low noise emission. Fan blading to have irregular pitch. If necessary, motor enclosure to be provided with special silenced ventilation.

Internal combustion engines (e.g., for driving quarry machinery)· Exhaust noise can be reduced by using amply dimensioned absorption and reflection silencers. Enclosure of the engine with ventilation system equipped with inlet and outlet silencers and ducts lined with sound-absorbing material (see Funke, 1977).

Fans: Small impeller clearances. Speed control. Main frequency can be modified by favourable choice of speed and number of blades. Acoustic insulation of casing walls. Enclosure of the whole fan and its drive. Silencers in the intake and exhaust ducts. Vibration-isolated mounting (see also VDI 2081).

Compressors, pumps: Vibration-isolated mounting in enclosed soundproofed rooms. Machines preferably separated from one another by partitions or with individual enclosures. Ventilation or air intake openings of such rooms should have louvred sound attenuators. Intake and outlet silencers for the compressors. Pressure pipelines to have sound-damping expansion joints and acoustically sealed wall inlets. Pressure release pipelines to have silencers. Additional sound insulation for compressed air pipelines to suppress rushing noises. Sound-insulated enclosed portable compressors represent the current "state of the art" for use in quarrying operations.

Crushers: Installation in enclosed sound-insulated buildings, preferably below floor level. Open side of building facing away from adjacent residents, if possible. Receiving hopper for stone tipped from trucks to have rubber lining and, preferably, a "cushioning" layer of material kept permanently in it. Soundproofed control cab for operating personnel.

Mills: Acoustic enclosure of the mill shell obstructs heat dissipation and makes inspection difficult, i. e., is not satisfactory. Boltless liner plates on rubber backing, or rubber liners, do not sufficiently reduce noise emission. A commonly employed and generally satisfactory solution is as follows: Whole grinding plant with all its noise sources (mill, drive, air separator, conveyors, elevators, filters, etc.) accommodated in a closed sound-insulated building with central control room for operators and with additional ventilation (see Funke, 1969 and 1973, Techn. Merkblatt, Bundesverband der Deutschen Kalkindustrie, 1975).

Planetary coolers Acoustic screening wall with ventilation openings; or fixed enclosure with ventilating fans, or movable enclosure around noisiest part of the cooler, provided (if necessary) with cooling fans if no water spraying system for cooling the tubes is installed. A more radical solution is to accommodate the whole kiln and planetary cooler in a closed sound-insulated building with air intake fans and with exhaust air outlets provided with sound attenuators (silencers) (see Funke, 1973).

Grate coolers: Sound insulation of the fan casings. Sound attenuators in the intake openings of the cooling air fans. Installing the fans below floor level and drawing in the cooling air through openings fitted with louvre-type sound attenuators.

Primary air fans: Sound-insulated enclosure of the fan. Sound attenuators in the air intakes.

Kiln hood (flame noise): Hood to be sealed as effectively as possible. Annular cover to close gap at outlet end seal. Sound insulation of fans near the kiln outlet and firing platform, just as for primary air fan.

Fuel oil pumps: Pumps installed in closed sound-insulated room or in an acoustic enclosure with insulated openings for the passage of pipes and controls (see VDI Code 2711).

Gas pressure reducing stations and pipelines: Closed sound-insulated buildings or acoustic enclosures. Sound-insulating lagging around pipelines. Sound attenuators in the pipes. Acoustic covers over valves.

Rotary kiln drives: Acoustic screening walls or acoustic enclosures with sound-absorbing linings, thermally insulated against kiln heat (if necessary) and with additional ventilation.

Material handling devices (conveyors, elevators): Acoustic enclosures for the drives. Shafts for bucket elevators built of concrete. Slow-running belt bucket elevators. Enclosure of squeaky chain conveyors or Redler conveyors. Belt conveyors: low belt speeds (below 1.5 m/sec); well-balanced rollers and efficient belt cleaners; low-noise bearings; if necessary, enclosure of entire belt in a closed duct or tunnel; rubber pad mountings for idlers in order to intercept structure-borne noise; large roller diameters; rubber-covered rollers not very effective. Entry points of belt conveyors into buildings (e. g., mill buildings) to be properly sealed against noise escape. If necessary, acoustic lock consisting of a sheet-metal tunnel, several metres in length, with sound-absorbing lining.

Screening machines: Screen decks of rubber or plastic. Flat screens instead of rotary ones. If necessary, screening machine to be completely enclosed, with access doors and inspection openings.

Portable machines: For noise control of internal combustion engines see above. Hydraulic pumps to be enclosed. Drivers' or operators' cabs to be separate, sound-insulated and on vibration-isolating mountings; adequate ventilation essential. Bodies of dump trucks to rubber-lined in order to reduce impact noise caused by stone or other material during loading.

Drilling machines (rock drills in quarry): Acoustic enclosure of drive units and hydraulic equipment, as on compressors used on construction sites. Noise-suppression covers and silencers for exhaust air outlets of hammer drills. Sound-insulated control cabs (see Funke, 1973).

Avoidance of impact sound emission: Reduction of impact intensity by reducing the heights of fall of bulk materials. Using construction materials with high internal damping capacity, e. g., rubber and plastics. Walls to be of rigid (non-vibrating) construction. Avoid vertical impact. Chutes and vertical ducts for solids to be provided with wear-resistant linings, or linings made of rubber or plastic in appropriate cases (no rapping necessary to assist movement). "Cushioning" to be provided by the handled material itself at transfer points, in hoppers, etc.

Reduction of rushing noise in pipes, etc.: Avoidance of small radii of curvature and abrupt cross-sectional changes in pipelines and ducts. Low circumferential velocities of fan impellers and rotors in electric motors. Avoidance of supercritical expansion ratios, e.g., due to pressure release at control valves.

Exhaust pipes and stacks: Should be installed in acoustically screened parts of buildings, but with due regard to possible sound reflection from adjacent wall surfaces. Installing sound attenuators which are unaffected by dirt or can easily be cleaned (see Funke, 1973). Cowls on stacks and chimneys to be designed as acoustic deflector cowls, if possible.

Machinery enclosures: Walls of acoustic enclosures should consist of heavy but flexible panels lined internally with sound-absorbing material. Adequate ventilation, demountability of the enclosure for repairs, and arrangements for operating and/or observing the machine to be provided. If hot gas fans are acoustically enclosed, the surface of the fan casing should additionally be heat-insulated. For information on design and performance of enclosures for noise control see VDI 2711.

2.3.4 Sound-insulated buildings

The requisite measures associated with the sound insulation of buildings, more particularly those for grinding mills, have been fully described in the literature (see Funke, 1969 and 1973; Techn. Merkblatt, Bundesverband der Deutschen Kalkindustrie, 1975). Table 14 gives the average sound reduction indexes (transmission losses) for the principal construction materials for walls and roofs as a function of their weight per unit area, for frequencies in the range from 100 to 3150 Hz. Table 15 gives sound reduction indexes for doors and windows. These values are applicable always on condition that the wall is completely closed and free from cracks. The sound insulation of walls with pores can be improved by plastering or rendering. On non-porous walls the plaster coating merely contributes to the sound deadening effect by its weight.
Walls which, for whatever reason, do not present a fully closed surface often have sound reduction indexes of less than 30 dB (see VDI 2571).
Sound levels in rooms can be lowered by up to 10 dB by means of sound-absorbing linings. However, such linings applied to walls and ceilings in mill buildings have only little effect (less than 3 dB reduction in level).

Table 14: Average sound reduction indexes for wall and floor construction materials

designation of building component	overall thickness mm	weight per unit area kg/m²	sound reduction index R dB
reinforced concrete slabs	4	95	37
	7	170	42
	10	230	47
	12	300	50
	15	350	52
	18	430	54
solid bricks, plastered both sides	7	170	42
	12	260	48
	24	480	55
sand-lime bricks, plastered both sides	12	260	48
	24	480	55

designation of building component	overall thickness mm	weight per unit area kg/m²	sound reduction index R dB
pumice concrete hollow blocks, plastered both sides	17	245	45
	25	290	50
pumice concrete solid blocks, plastered both sides	12	145	14
	12	205	45
gas concrete floor slabs	240	160	45
prestressed concrete hollow planks	120	220	49
pumice concrete hollow plans	120	185	49
1 mm sheet steel (flat)	1	8	25
1 mm sheet steel (trapezoidal section)	45	11	26
1 mm sheet steel (double trapezoidal section)	190	22	35
1 mm sheet steel (trapezoidal) with mineral fibre boards	120		32
1 mm sheet steel (double trapezoidal) with mineral fibre boarde	190		41
two 1.5 mm sheet steel panels with rigid foam plastic filling	60		40
roof covering with 2.4 cm wood particle boards and two layers of roofing felt	3	19	30
timber roof with bracing members (25 mm thick)	115	14.5	27
corrugated asbestos cement (6 mm)	55	12.5	19
corrugated asbestos cement (6 mm) with mineral wool boards	330		28
wood particle boards (chipboards)	36	20	28
wood particle boards (chipboards)	16	10	24
glazing			
building glass	3.5	9	30
thick sheet glass	6	14	32
laminated safety glass	8	19	35
solar heat rejecting glazing with two 4 mm panes	20	19	35
glass building blocks DIN 18175			
size 11.5 cm × 24 cm	5	80	37
size 24 cm × 24 cm	8	100	39
size 30 cm × 30 cm	10	125	41

Table 15: Average sound reduction indexes for doors and windows

	sound reduction index dB
ordinary inner door (timber)	15–22
heavy door with rebated jambs (timber)	25–30
special sound-attenuating door (timber)	30–40
double-leaf sheet-steel special door with rebated jambs	35–50
two ordinary timer doors, one behind the other	35–45
ordinary window	20–25
composite window	25–30
box-type double window	30–40

2.3.5 Ventilation of buildings

All buildings which have to be of closed construction for purposes of noise control will require suitable ventilation arrangements more particularly if they accommodate machinery which gives off heat.
The **heat flow** to be removed from the building with the flow of spent ventilating air is calculated as the difference between the heat input (e. g., the thermal equivalent of the electric power absorbed by the mill drive motors) and the heat removed through other media (e. g., heat content of the mill product discharged from the building). The required cooling air flow rate can be calculated according to Funke/Keienburg/Sillem (1975).
For buildings of the size nowadays usually employed, an approximate guiding value is between 2.8 and 5.5 m^3 per kW of installed drive power, with between 5 and 15 **air changes** per hour in order to keep the temperature in the upper part of the building below about 40° C in summer (see Funke, 1973).
For mill buildings constructed in recent years the following three systems of cooling have been found effective:

(1) Natural ventilation with air outlets on the roof (Fig. 18). For maximum permissible air velocities of 1–2 m/sec the cross-sectional areas of the openings have to be quite large. Noise emission from them can be reduced by 20–40 dB by means of labyrinth-type or louvre-type attenuators.
(2) Forced-draught artificial ventilation by fans which force cooling air through ducts into the mill building. The fans are equipped with air intake sound attenuators. The heated air rises in the building and escapes into the atmosphere through outlets which likewise have attenuators (Fig. 19).
(3) Induced-draught artificial ventilation by axial-flow fans mounted on the roof and fitted with sound attenuators on their outlets. The air for cooling the interior of the building enters through openings at the base, these likewise being provided with attenuators (Fig. 20).

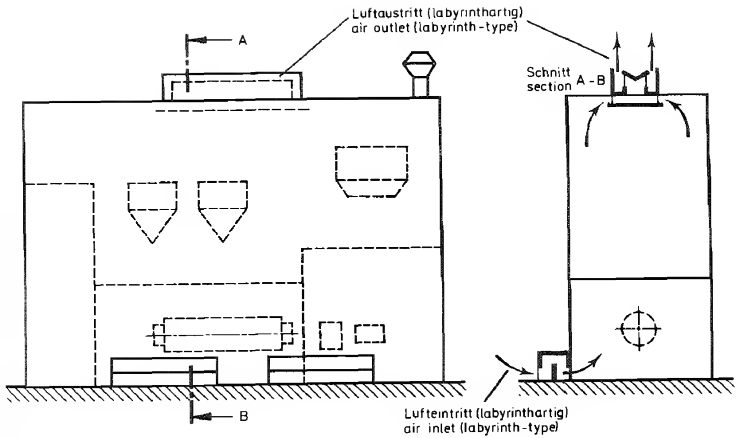


Fig. 18: Natural ventilation of a mill house

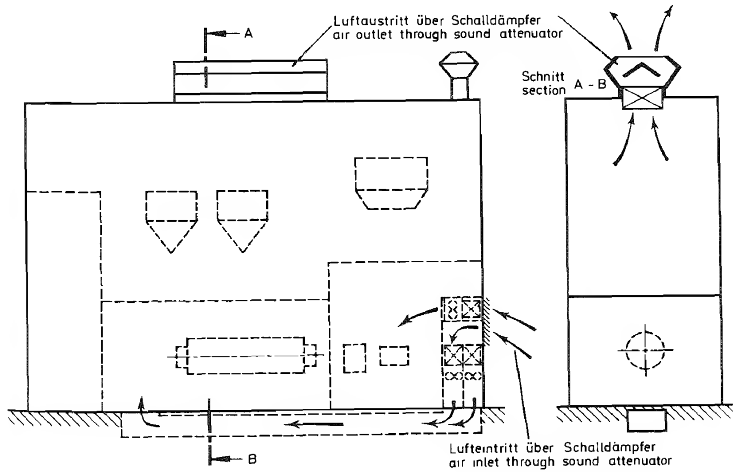


Fig. 19: Forced-draught ventilation of a mill building: air blown in by fans

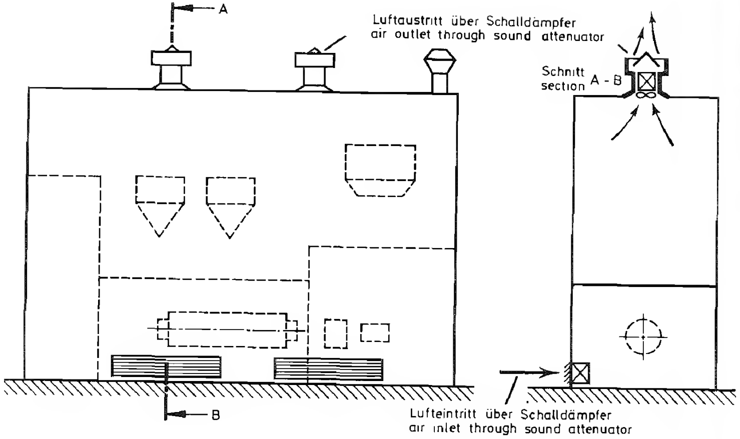


Fig. 20: Induced-draught ventilation of a mill building: air extracted by fans

Air inlet and outlet openings, as well as openings in intermediate floors and partitions should be so dimensioned and located that thorough ventilation of the mill room, drive machinery room and any other parts of the building from which heat has to be removed is duly ensured.
In determining the dimensions of the air intake openings or the capacity of the fans, it is necessary to take account also of the amounts of air extracted from the plant by its dust collecting system and any cooling air that may additionally be required for the mill drive.
The air inlets and outlets of the building, as well as the intake and discharge openings of the fans, should be provided with suitable **sound attenuators** (silencers) (see Funke, 1973).
Air extractors with wind baffles so designed as to produce additional suction at the outlet, thus inducing a draught of air, have proved very suitable for the removal of warm air from buildings.

2.4 Noise in the working environment

See VGB 121, 1974; VDI 2058, Sheet 2.
Prolonged exposure of people to excessively high noise levels is liable to damage their hearing.
The German regulations for the prevention of noise injury, embodied in the document known as VGB (1974), make it obligatory upon industrial undertakings to take appropriate measures for noise control in their employees' working

environment. If an assessment level of 85 dBA is exceeded, the employer is required to provide his workers with personal protective devices (e.g., ear muffs); for a level of 90 dBA or higher the workers are obliged to wear these devices.
Noise zones in which the assessment level of 90 dBA (average value per working shift) is reached or exceeded must be indicated by means of special symbols displayed in the factory.
In clause 15 of the statutory regulations relating to places of work (Verordnung über Arbeitsstätten, 1975) the following maximum values for the reference noise level in the working environment are laid down:

(1)	for predominantly mental work	55 dBA
(2)	for simple or predominantly mechanical office work and similar activities	70 dBA
(3)	for other activities	85 dBA.

In off-duty staff rooms, rest rooms and first-aid rooms the assessment level must not exceed 55 dBA. For calculating the assessment level, any extraneous noise from outside the building should in principle be taken into account.

2.4.1 Protective measures

Besides the basic general rule of only using machines designed and/or acoustically insulated or enclosed to as to emit the least possible noise, there often exist opportunities of reducing objectionable or harmful noise levels in a working environment by **organizational modifications** in the factory. The primary object should be to keep down to a minimum the number of persons engaged in duties in very noisy surroundings. The following measures can help to achieve this:

- Ancillary activities which do not necessarily have to take place in high-noise areas – e.g., cleaning, maintenance and repair of removable components, packing operations, production preparations, etc. – should be carried out in quieter surroundings.
- Noisy machines should be accommodated in different rooms from quieter ones. Special acoustic enclosures for machinery with particularly high noise emission levels.
- Remote control of sound-insulated or enclosed machinery from a control centre.
- Reduction of exposure time to high noise levels by changes of personnel at certain intervals. Periods of duty under high-noise conditions to alternate with periods in quieter surroundings where the assessment level is not above 75 dBA.

So far as process engineering requirements allow, high-noise activities should not take place in the evening and night hours. Furthermore, at such times, doors and windows of buildings containing noisy machinery or equipment should be kept closed, if possible, in order to reduce noise emission. Also, if starting of machinery temporarily causes high noise levels, this should be avoided at such times.

2.4.2 Personal protection against noise

See VDI 2560.
Personal protective measures include more particularly:

- Acoustic cabins and booths for machine operators or supervisors. Such enclosures should achieve a sound level reduction of between 20 and 30 dB and be adequately ventilated; air conditioning may also be necessary. They are often of demountable construction, so that they can conveniently be removed and reassembled where and when required. Suppliers of acoustic cabins, etc. are listed in the publication LSI 01-200 (1977).
- Screens or partial enclosures protecting the immediate working environment. Such measures achieve sound level reductions of less than 15 dB.
- Individual ear protectors: sound-absorbing cotton wool, ear plugs, ear muffs, acoustic helmets, etc. VDI 2560 "Personal hearing protection" gives information on all matters relating to the subject. A review of types, a list of suppliers and particulars of equipment are given in LSI 01-830 (1978).

2.5 Guarantee conditions for noise control measures

Suggestions for tendering, ordering and guarantee agreements are given in Appendix D of VDI 2570 "Noise abatement in factories". Further information on these matters is contained in the Technical Information Note "Noise abatement in the works of the lime industry" (Technisches Merkblatt, Bundesverband der Deutschen Kalkindustrie, 1975) and the Code SEB 905003-63 (1963).

3 Ground vibrations due to blasting

3.1 Origin and properties of ground vibrations

See Thum (1978).
The energy which is released in blasting operations carried out in cement works' quarries is consumed more particularly for loosening, dislodging, breaking up and hurling away the rock. Part of this energy is inevitably transmitted to the surrounding rock and is propagated through it in the form of elastic waves which are experienced as "ground vibrations" or "shocks".
Besides the detonation impact and the pressure developed by the gases formed in the explosion, other phenomena may play a part, e. g., sudden pressure relief in the surrounding rock, the initiation of additional stresses in it, and the impact due to the fall of dislodged rock masses.
The **velocity of propagation** of the waves through the subsoil will depend on the elastic properties thereof (Table 16).
The actual ground motion due to blasting vibrations lasts for about one second. The form and behaviour of the oscillations are governed by the type and size of the explosive charges, the transmitting and damping properties of the soil, and the distance from the site of blasting. It attains its largest value at the surface of the ground because here the soil particles have the greatest freedom to oscillate.
A distinction is to be drawn between "body waves" (a seismological term comprising pressure waves and shear waves), which are of major importance over short distances (up to about 100 m) from the explosion site, and "surface waves", which travel over appreciably longer distances before dying away. Depending on the soil conditions, these respective types of wave may have different amplitudes, frequencies and propagation velocities, so that they will not necessarily arrive simultaneously at the point of observation.
The frequencies of the vibrations depend more particularly on the nature of the subsoil. Compact and solid rock has only a low damping capacity, and the frequencies are relatively high (20 to 60 Hz). A subsoil which is less strong or is of a more yielding or plastic nature will develop a greater damping effect, so that the vibration frequencies are lower (5 to 20 Hz).

Table 16: Propagation velocity of elastic waves in the subsoil

type of soil or rock	propagation velocity
sand, gravel, loam in ground-water	1000–1500
moraine, slate, soft limestone	2000–3000
hard limestone, quartzitic sandstone, gneiss, granite, diabase	4500–6000

3.2 Measurement and assessment of ground vibrations

See DIN 4150, Preliminary Standard.
The most suitable measurable quantity for recording the ground vibration phenomena in a seismogram is the velocity of vibration. The following relationships are valid:

v = velocity of vibration (mm/sec) $= 2\pi \cdot f \cdot a$
b = acceleration of vibration (mm/sec^2) $= 4\pi \cdot f^2 \cdot a$,
where f is the frequency (sec^{-1}) and a the amplitude (mm).

According to DIN 4150, Part 3, "Vibrations in structural engineering", the criterion for assessing the effect of ground vibrations on buildings is provided by the largest peak value of the resultant vibration velocity occurring at the foundation of the building. More particularly, this is the resultant of the three mutually perpendicular velocity components v_x, v_y and v_z:

$$\hat{v}_R = \sqrt{v^2_x\,(t) + v^2_y\,(t) + v^2_z\,(t)}\ \max.$$

The maximum values of the three velocity components usually do not occur simultaneously. However, as a simplifying approximation, an "equivalent resultant" is calculated from the three maximum values:

$$v_{eqR} = \sqrt{\hat{v}^2_x + \hat{v}^2_y + \hat{v}^2_z}.$$

This approximation is on the safe side, as v_{eqR} is always at least equal to, or larger than, the actual vibration velocity resultant $\hat{v}_R$.
For the measurement and assessment of the effects of ground vibrations due to blasting on **buildings** (see DIN 4150, Part 3), the vibration pickups (transducers) for the three mutually perpendicular velocity components are installed in or on the foundation of the building or otherwise in the wall as close above ground level as possible and preferably on the side facing the blasting site.
Measurements for assessing the effects of these vibrations upon **human beings** inside buildings (see DIN 4150, Part 2) are usually performed with the vertical vibration pickup installed on the floor in the middle of the room under investigation. The horizontal components should be measured by pickups installed in or close to the walls of the room (e.g., in door or window recesses in these walls). The largest value measured in the vertical or the horizontal direction is to be adopted as the significant quantity for the evaluation. The effect of ground vibrations on people is assessed with the so-called "perceived strength", a dimensionless value which is calculated from the peak value of the vibration velocity and the frequency of the vibration under assessment.
The values calculated in this way are compared with the guide values for the effects of blasting vibrations on buildings and on people given in DIN 4150, Parts 2 and 3. These guide values are graded according to the type of building, its structural condition and the region in which the building is located.

3.3 Prediction of ground vibration intensities

Various factors significantly affect the intensity of the vibrations produced by underground explosions due to blasting, e.g., the charge fired per firing interval and the overall charge of explosive, the stemming, the burden and the blasthole spacing, the method of firing, the nature of the surrounding rock, the soil conditions along the transmission path of the vibration waves, the type of building, its foundations, etc. Although a considerable amount of experience has been gained and much information relating to these matters has been collected in recent years, it is not yet possible to make sufficiently accurate predictions of the intensity of the vibrations. As a rule, it is necessary to rely on measurements of the vibrations that actually occur.
In the immediate vicinity of quarry blasting sites (within a radius of a few hundred metres) the following approximate relationship has often proved reasonably valid:

$$v = C_1 \cdot \frac{\sqrt{L}}{d},$$

where:

v = vibration velocity in mm/sec
L = explosive charge per firing interval in kg
d = distance from blasting site to point of measurement in m
C_1 = proportionality constant.

The following variants of the above formula give more accurate results in some cases:

$$v \sim C_2 \cdot \frac{L^{3/4}}{d^2} \text{ or } v \sim C_3 \frac{L^{2/3}}{d}.$$

When the proportionality constant C_1 (or C_2 or C_3) has been determined in a given situation, preferably from measurements associated with several blasting operations, the vibration velocity for charges of various sizes and at various distances from the point of observation can be estimated with the aid of the above formulas.
For preliminary approximate assessments a factor K is substituted in lieu of the proportionality constant. Some typical values for this factor are as follows:

type of rock	factor K
shell limestone	120
limestone	80
basalt	60
marl	50.

The magnitude of K decreases with increasing distance, because the probability of cracks and fissures in the rock increases. Sometimes, to be on the safe side, a value of 150 is adopted for K in calculating the maximum permissible explosive charge.

3.4 Ways and means of reducing the vibrations

Some possible measures for reducing the ground vibrations associated with blasting in quarries are listed below. Which of these measures can and should be adopted will depend on the aspects presented by each individual case.

3.4.1 Measures relating to quarrying procedure.

- changing the direction of quarrying advance;
- reducing the height of the faces;
- not blasting below ground-water and not employing bottom initiation;
- keeping certain minimum distances between blasting and neighbouring property liable to be affected;
- instead of blasting, using rippers to loosen and break up the rock (possibly with some supplementary blasting).

3.4.2 Measures relating to blasting technique:

- using large-hole blasting technique;
- using electric millisecond delay detonators (20 or 30 ms);
- limiting the quantity of explosive per hole and per blast;
- firing each hole with one firing interval;
- reducing the number of holes or the amount of rock pile per blast;
- reducing the burden and blasthole spacing;
- changing over to smaller blasthole diameters;
- using a higher-strength explosive in the bottom part of the holes;
- employing special blasting methods with horizontal holes;
- subdividing each blasthole into several charges separated by intermediate stemming and fired in succession by delay detonators;
- one blasthole to be given a smaller burden and smaller charge than the other holes and to be fired first.

The above-mentioned measures cannot always be applied. Also, the degree of success achieved with them is likely to vary greatly from one set of quarrying conditions to another. Other measures are described in the literature (see Thum, 1978).

3.5 Other environmentally objectionable emissions from quarrying

See Funke (1977). When the explosive charges are fired, **dust** may be emitted from the blastholes. Also, with dry rock a cloud of dust will be thrown up by heavily fragmented rock crashing down onto the quarry floor.
Danger from flying fragments of rock may occur in the following circumstances:

- if the burden is too small or if a borehole "wanders off course" during drilling and terminates too close to the face;
- inadvertent detonation of misfired charges (e.g., when toes which have remained standing have to be removed by secondary drilling and blasting);
- inexpert drilling and charging of secondary holes for the fragmentation of toes and boulders.

Large boulders which cannot be directly accepted by the crusher have to be reduced in size. This can be done by blasting or by mechanical means, e.g., by means of special hammers or by the drop-ball method.
One secondary blasting method for dealing with boulders is "mud-capping", in which the explosive charge is applied to the surface of the boulder. This is not a good method because it involves a higher risk of flying rock fragments, besides being noisy. Instead, holes should be drilled into the boulder and be charged with only just enough explosive to split it. A fairly low-strength explosive should be used for the purpose, and the detonators and detonating fuse adequately covered.
Hydraulically powered rock breakers mounted on excavators are often of only rather limited capacity. Besides, it has hitherto not proved possible satisfactorily to reduce the high levels of noise emitted by these devices with their hammering action, but further development is likely to achieve improvements.

Secondary blasting operations should preferably be carried out in parts of the quarry which are acoustically screened from adjacent residential buildings. Besides, they should be confined to certain hours of the day, with suitable intervals of quiet between operations.
When the explosive charge in a borehole is detonated, the detonation shock not only sets up waves in the surrounding rock, but also produces vibrations in the air which are emitted as **airborne sound**. Considerable sound pressure levels are produced: for example, under free sound propagation conditions, a level of 110 dBA was measured at a distance of about 110–120 m from a large-hole blast. Even higher levels may occur with secondary blasting, especially if the mud-capping method is used. The noise emission from lare-hole blasting operations can be reduced more particularly by appropriate technical measures, e.g., by bottom initiation of the holes (with electric detonators), reducing the charge per hole, stemming the top part of the holes, etc. With these precautions, the blast does not produce a sharp bang, but a dull muffled noise because detonation is achieved under completely enclosed conditions and the gases evolved by the explosion are released into the atmosphere only after dislodgment of the rock burden, by which time they have largely expanded. With toe drilling, the charges are covered with a 60–80 cm thick layer of sand, which not only muffles the detonation, but also reduces the danger from flying fragments of rock.
With large-hole blasting, the maximum of the noise emitted is in the low frequency range, between about 50 and 500 Hz. On the other hand, secondary blasting often produces a very objectionable bang with its maximum in the strongly audible frequency range of between about 500 and 2000 Hz (see Funke, 1978).
It is important not to carry out blasting at night, nor during the daily rest periods in the morning, afternoon and evening. As far as possible, blasting should be programmed to take place at certain times of the day and the neighbouring residents be informed of these. This is desirable because sudden unexpected noise is experienced as especially objectionable.

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II. Industrial safety

1 Accident prevention regulations

In the Federal Republic of Germany the responsibility for drawing up accident prevention regulations for the protection of people at work has been entrusted to the so-called Berufsgenossenschaften (employers' liability insurance associations). These regulations are embodied in Codes (designated as UVV) and have statutory status: compliance with them is compulsory.

1.1 Employer's obligations

Under Section I of UVV 1.1 "General regulations", the employer is required to fulfil the following obligations for the prevention of accidents:

- Make arrangements and take precautions conforming to the requires of the UVV Codes and of other generally acknowledged rules of safety engineering and occupational medicine.
- Stop any installations in which a defect has developed which constitutes an otherwise non-preventable hazard to the employees.
- Make available personal protective equipment if, because of technical operations, it cannot be ruled out that the employees will be exposed to accident or health hazards. Such equipment is to be kept in proper working order.
- When having work carried out under contract: inform the contractor in writing of his obligation to comply with the general regulations for the prevention of accidents to people at work.
- When having work carried out under contract: appointing a person to co-ordinate the activities. This person must be given authority to issue instructions to the contractor and the latter's employees.
- Display the text of the relevant UVV Codes in a suitably accessible place.
- Instruct and inform the employees as to the dangers associated with their work and as to the measures for the prevention thereof. This should be done before they start the job and afterwards at appropriate intervals, but at least once a year.
- Issue copies of the accident prevention regulations to the persons entrusted with their enforcement within their appointed spheres of duty.
- Encourage the employees to participate in accident prevention (e.g., by attendance of training courses on industrial safety).
- Appoint safety officers as envisaged in RVO, clause 719, and give them adequate opportunity to carry out their duties.
- On request, give factory inspection officials facilities to inspect the works.
- Inform the Berufsgenossenschaft (employers' liability insurance association) that the required safety measures have been complied with.
- Provide all information required in connection with accident prevention arrangements.
- Give immediate notice in writing of any accident prevention obligations transferred by the employer to others.

- Clearly define the spheres of responsibility of the supervisory personnel appointed.
- See to it that the obligations as to accident prevention and co-ordination of activities are duly fulfilled.

1.2 Obligations of the employees

Under Section II of UVV 1.1, the employees must fulfil the following obligations:

- Support all measures for the promotion of industrial safety.
- Follow instructions issued by the employer with a view to accident prevention, except under circumstances where they are evidently unnecessary.
- Use the protective equipment made available.
- Not carry out instructions at variance with safety requirements.
- Use installations and appliances only for the purposes intended by the employer.
- Correct any faults presenting a safety hazard, or report them to their superiors, without delay.
- Use installations, appliances and materials, and enter installations, only when authorized to do so.

1.3 Industrial installations and regulations

Section III of UVV 1.1 lays down requirements concerning the place of work in buildings, in the open air and on traffic routes, as well as concerning doors, gateways, escape routes, emergency exits, escalators and loading bays. In addition, it contains regulations for protection of personnel against falling from dangerous heights, for protection against injury by falling objects, for storage and stacking arrangements, for the wearing of clothing and ornaments, and for the carrying of tools and other objects.
Of great importance are the sections on hazardous activities, admittance to dangerous areas, consumption of alcohol, testing of equipment and machinery, and precautions against fire and explosions.
In Section IV the UVV 1.1 Code deals with medical precautions and the necessary medical checks and examinations.

1.4 Special accident prevention regulations for cement works

In an Information Note (Merkblatt) MuU1 issued by the Verein Deutscher Zementwerke (German Cement Works' Association), January 1978, the principal UVV Codes relating to cement works are set forth with a view to providing those responsible for industrial safety with an aid to the fulfilment of their duties. These texts are subdivided according to the various sections of the industry, such as the quarry, the crushing and grinding plants, the kiln plant, the packing plant, the fitters' and motor vehicle repair shop, the electrical repair shop, carpenter's shop,

construction department, laboratory, etc. They comprise the principal relevant UVV Codes, together with other directives and information notes, for distribution to the foremen and gangers. In addition, the Appendix contains a model text of an agreement for the transfer of obligations to third parties and also a model text of an agreement of acceptance of obligations by suppliers of technical equipment and services.

2 Promotion of safety in cement works

Fulfilment of the obligations enumerated in Sections 1.1 and 1.2, above, on the part of the employer and of the employees does not automatically ensure safety-conscious working in the factories and other industrial installations concerned. To achieve this it is necessary additionally to provide information, motivation and other inducements. Some examples will now be given:

Information and instruction

- Displaying information on accidents that have occurred in the various sections of the works and in the works as a whole.
- Displaying information (tables, graphs) showing the number of accident-free days since the last notifiable accident.
- Displaying posters stating "dos and don'ts", showing how accidents occur, etc. These should cover not only accidents in the works itself, but also road accidents ("theme of the month").
- Displaying safety information notes issued by the German Cement Works' Association showing typical accidents; or similar information on accidents that have actually occurred in the works.
- Instructing the employees, especially those newly recruited, as to the dangers associated with their work and the safety rules they should obey.
- Circulars on industrial safety matters to the employees and their families.
- Information on accidents should be reported at works meetings, possibly backed up by the showing of films or slides with spoken commentary.
- Special instruction of employees when commissioning and starting up new installations.
- Instruction on the hazards due to non-use of personal protective equipment (e.g., wearing of safety helmets, safety footwear, ear protectors, etc.).
- Instruction on the problem of safeguarding installations under repair against unauthorized or inadvertent switching-on.
- Discussion of themes relating to industrial safety, analysing the causes of accidents, at section engineers', foremen's and works management meetings.
- Collaboration with the planning department and the supplying firms (suppliers of machinery, etc.) with a view to achieving optimum safety conditions both in the normal running and in the maintenance and repair of plant.

Motivation to safety-consciousness

- Positive motivation by encouragement and persuasion is preferable to negative motivation by scaring.
- Special posters encouraging safety-conscious behaviour are more effective than general posters warning against accidents.
- Key personnel can give a good example by safety-conscious behaviour.
- Constant reminders that preserving one's health is the greatest benefit.
- Positive response to safety-consciousness displayed by employees (commendation, thanks).
- Personal conversations between key personnel and their subordinates on the meaning and purpose of safety measures.
- Safety-consciousness of senior personnel: establishing the right balance between safety and productivity.
- Co-operation between the safety officers, the senior works personnel and the management.

Bonus systems, competitions and other material incentives

- Individual bonuses
- Collective bonuses (to teams, gangs, etc.)

} incentive rewards and penalty deductions

- Special bonuses
- Safety competitions within the works
- Safety competitions between different cement works
- Savings due to reductions on industrial accident insurance premiums may be distributed among the works' employees.

Information on bonus systems is given in Information Note (Merkblatt) MiB2 issued by the German Cement Works' Association.

Other measures (examples).

- Safety "stickers" on safety helmets, motor vehicles, etc.
- Issuing plastic wallets, key rings with tabs, etc. with safety-promoting inscriptions.
- Free testing of the lights and exhaust gas toxicity of employees' motor cars.
- Safety checks on cars and motor cycles of all employees, in collaboration with the police.
- Check lists for instructing newly appointed employees: these lists should be signed both by those receiving and by those giving instruction and be filed with each employee's documents.
- Safety check lists for regular inspection of specific items of machinery or equipment (e.g., on a monthly basis).
- Accident statistics to be compiled in the individual works and by the German Cement Works' Association for the industry as a whole.

3 Safety rules

Safety rules which are drawn up for use in individual works – supplementary to the UVV Codes already mentioned – should be concise and written in simple language (which foreign workers, if any, can also understand). Where possible, they should contain drawings and diagrams illustrating the significance of the principal accident prevention measures. The message they convey should be aimed at all personnel: safety officers, foremen, operatives engaged on duties in the various parts of the works, and any employees of outside firms whom it may concern.
Some examples of generally applicable safety rules which should be complied with by all works personnel will now be given:

(1) Wear protective clothing as and when required (safety helmets, footwear, gloves, goggles, masks).
(2) Keep your place of work and your tools neat and tidy. Do not use any damaged or defective tools, instruments or other equipment.
(3) Take proper care when dealing with flammable or caustic substances.
(4) Do not remove or detach any protective devices or safety appliances unless authorized to do so. Keep them in good working order. Do not start any machines on which guards, screens or other protective devices are missing.
(5) Do not start a machine until you have satisfied yourself that it is in proper working order and that there is no danger to any person.
(6) Never carry out repairs on a machine while it is running.
(7) When carrying out repairs, make sure that the machine or equipment cannot be started or switched on inadvertently. This can be ensured by switching off the current and displaying a notice saying "Do not switch on! Repairs in progress!" or by locking the switch so that it cannot be operated.
(8) Clean and lubricate moving parts of machinery only if suitable protective devices are provided.
(9) Report any damage to machinery, including machinery of which you are not in charge yourself.
(10) Take care when handling burning and soldering equipment; immediately repair or replace any defective parts of such equipment.
(11) Hands off electrical machines and appliances! Do not try to carry out repairs to them yourself: leave that to electricians.
(12) Place ladders, scaffolding and working platforms securely in position so that they will not fall or collapse.
(13) When carrying out erection, building or demolition work, prevent access by unauthorized persons.
(14) Make sure that pits, trenches, etc. are properly safe.
(15) Secure yourself and any loose objects against falling from heights.
(16) Do not stand under loads being lifted by cranes, etc.
(17) Take care when crossing motor vehicle traffic routes or railway lines. Only use the public railway crossings.

(18) Do not allow unauthorized persons to travel as passengers in road vehicles, locomotives or goods waggons.
(19) Never ride on conveyor belts or on loads handled by cranes or other lifting appliances.
(20) Avoid consuming alcohol before and during work.
(21) Immediately report any accident. Giving aid to victims of accidents is your obvious human duty. Have any injuries, even minor ones, immediately attended to.

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J. Maintenance and wear

By B. Kohlhaas

I. Maintenance

1 General

The maintenance and servicing of industrial installations is of major importance if serious production losses are to be avoided and the cost of manufacture is to be kept as low as possible. It is therefore essential that plant and equipment preserve their efficiency. From this point of view, maintenance is in the interests not only of the individual manufacturer, but of the economy as a whole.
A high level of operational availability of machinery and other facilities must be ensured in order to enable the planned output of the production process always to be attained at lowest cost. Unscheduled shutdowns should be prevented as far as possible. Maintenance should be organized with reference to three guiding principles:

(1) Deciding what is to be inspected and at what intervals of time. An interesting computer-controlled system has been devised for the purpose and is described in the Appendix to this section (Kunze, 1978).
(2) Careful planning and execution of the maintenance work by the works department responsible for these duties.
(3) Preventive maintenance (see Weislehner, 1976, and Hochdahl, 1976)

The basis for maintenance work should of course be provided by the operating and servicing instructions issued by the suppliers of the various items of plant and equipment.

Three schemes should be drawn up.

- The first scheme should indicate, in a convenient and clearly visualizable manner, what machines or appliances have to be inspected at regular intervals.

Damage or faults can thus be detected in good time and measures for effectively remedying them be initiated.

- In the second scheme it should be stated how the necessary repairs, servicing operations, etc. are to be carried out, i.e., what materials, tools, spare parts, etc. are required and which personnel (number and technical skills) are needed for doing the job.
  In this way it is ensured that the right men for the job are available at the appropriate time and that the special spares are ready to hand (spare parts planning!).
  If possible, the estimated lengths of time required for carrying out this maintenance work should be included in the scheme, so that the loss of production likely to occur in the event of a shutdown period of the plant or parts thereof can be gauged in advance.
- The third scheme should indicate what work should be carried out during scheduled or unscheduled shutdown periods, e.g., lining brickwork construction, changing the liner plates in grinding mills, changing the grinding media charge, etc.

Obviously, not all possible details of plant maintenance can be comprised in a preventive maintenance scheme. A carefully considered choice must therefore be made as to what can and what cannot be included.
It is of course necessary, after execution of the work, to keep accurate records of repairs and replacements, with precise details of what parts were affected and how much time was spent on putting the trouble right. The information obtained in this way is not only useful as a basis for future planning, but also provides reliable guidance in calculating the cost of maintenance.
In the day-to-day running of any major plant it is not possible to rule out the sudden and unexpected occurrence of technical faults. These have to be remedied at once in order to keep downtime and loss of production to a minimum. Any such event must of course be carefully recorded.

2 Spares and renewable parts planning

In order to ensure efficient execution of repairs it is necessary to pursue a forward-looking policy of spare parts inventory and procurement of wearing parts which have to be renewed from time to time.
Prolonged shutdowns of production facilities or other vital installations must be avoided as much as possible. For this it is essential to have the requisite parts and components available in sufficient quantity and quality as and when they are needed. The decision as to what parts, and in what quantities, are to be kept in stock will have to be based on recommendations from the suppliers of the relevant machinery and equipment and on the works management's own experience.
In this connection a distinction should be drawn between "spare parts" and "renewable parts".

"Spare parts" comprise all those parts which may fracture or become faulty in some other way as a result of long service during normal operation of the plant (e.g., springs, bolts, seals, ball bearings, other machinery components, etc.).
The term "renewable parts" or "wearing parts" relates to those parts which, in consequence of intimate contact with the production materials or auxiliary materials, are subject to natural wear (e.g., grinding media, liner plates, refractory materials, linings of chutes, etc.).
A distinction is also to be drawn between commercially available standard parts and special parts which can best be procured from the manufacturers of the machinery concerned.
All spare parts and renewable parts should be kept in a properly equipped store and be carefully catalogued and controlled.

3 Determining the cost of maintenance

The expenditure in respect of wages and materials for repairs and maintenance work executed should be carefully recorded, as it constitutes a significant item of production costing. The works management should at all times have a general view of such expenditure, so as to be able to take appropriate measures if it shows signs of getting out of hand. The expenses incurred should not, however, be recorded in a general overall way, but should be itemized in detail according to the various production departments concerned and be split up into wage costs and material costs respectively.
This approach will enable the man in charge of maintenance to detect any unusual trends, to look for the causes of these and, if necessary, to take remedial action
Although the problems of costing – of whatever kind – belong to the sphere of industrial management, it is nevertheless important that the plant engineer should be alive to the cost aspect, because the two aspects of plant operation – the technical and the economic – must always be sensibly geared to each other.

Appendix

WARTAS – a computerized system of plant inspection and maintenance control

In connection with the ordering of new plants, or parts thereof, it has become common practice to require a schedule of all maintenance and servicing work required. The reasons for this are as follows:

- With the introduction of new technology, the relative cost of mechanical and electrical engineering is steadily rising and often exceeds 50% of the overall capital cost of the factory or plant.
- The maintenance instructions embodied in manuals, etc. vary from one machinery manufacturer to another and, in many cases, relate to machines and appliances occurring in many different parts of the plant.

- For example, the complete operating documentation of a cement works with a clinker output of more than 3000 t/day comprises some 25 standard lever-arch or box files. Preparing the maintenance programs from this mass of machine manufacturers' literature requires the services of five experienced maintenance experts and organization experts.

In order to cope with these requirements, as well as with others likely to arise later, a future-oriented organization technique for industrial plants has been developed: the computerized maintenance system called WARTAS (Fig. 1).

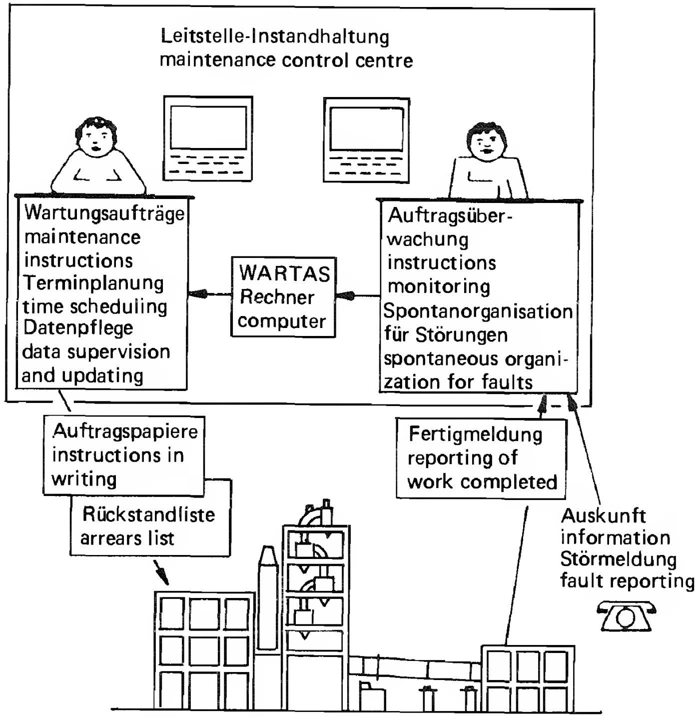

Fig. 1: WARTAS automated maintenance time scheduling system (KHD Humboldt Wedag AG)

This technique offers the following facilities and advantages:

- Complete and appropriately timed output of information on all maintenance work to be performed.
- Key personnel largely relieved of routine duties, thanks to the timing and information system employed.
- By spreading and equalizing the work load of the maintenance personnel the system increases their productivity.
- The manufacturers' experience can be advantageously utilized in the preparation of the master data. Specialist engineers can prepare the data for the particular plant concerned, in readiness for their application in practice.
- The system yields daily updated information and reports.
- A computer-controlled analysis of potential trouble spots is associated with the monitoring of important operations.
- The optimization system for timing the proposed shutdowns for maintenance is specially geared to the needs of industrial plants.
- The personnel in charge of allocating the tasks and duties in connection with maintenance are provided with a ready-made organization for dealing with out-of-the-ordinary occurrences.

Special aspects of the new organization technique will now be briefly considered:

(1) Organization of maintenance:

The important feature of the organization scheme is the maintenance control centre (Fig. 2) with the following range of activities

- Allocation and monitoring of the inspection and maintenance jobs put out for the next working shift by the electronic data processing system.
- Acting upon the suggestions or instructions issued by the electronic processing system in the event of a fault or breakdown.
- Information service, with video screen backing, for the maintenance gangs and works management.
- Evaluation of the trouble spot analysis and modification of maintenance instructions and servicing intervals on the basis thereof.

The system does not require any major changes in the existing organization of foremen's duties, but aims rather at relieving key personnel of mere routine work.

(2) Timing procedure:

A number of methods, depending on the customer's preference and requirements, are available for the time scheduling of the maintenance operations (Fig. 3):

- The degree of freedom in timing the operations will depend on plant operating conditions (e.g., waiting for shutdown).
- The frequency with which any particular servicing job is carried out will depend on its nature and demands.

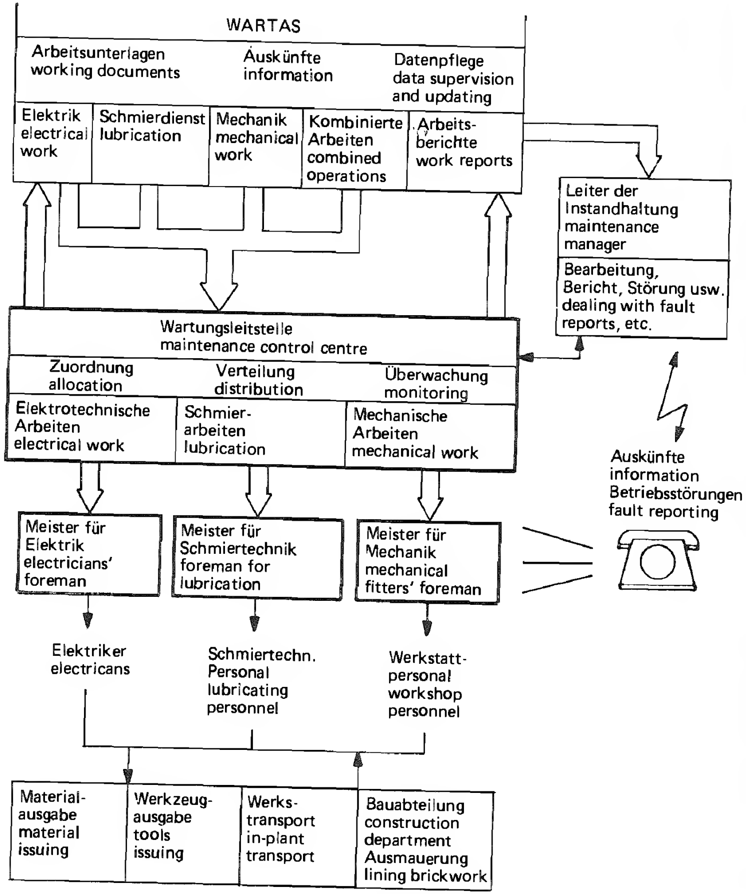

Fig. 2: Maintenance information flow diagram (KHD Humboldt Wedag AG)

time scheduling	operating condition	execution of work	job monitoring
– according to calender days	– job is of such importance as to justify shutdown	– regularly	– acknowledgment of completion required
– according to plant operating hours	– job can be carried out at next shutdown	– one-off	– no acknowledgment signal to computer required
– at next shutdown	– job can be carried out while plant is running	– one-off, followed by subsequent work at regular intervals (one oil change, special servicing, servicing after an inspection)	– limiting level for permissible number of arrears

Fig. 3: Procedure for control of maintenance jobs (KHD Humboldt Wedag AG)

- The timing of operations distinguishes between "periodic" and "dynamic" ones.
- The closeness of monitoring characterizes the importance of maintenance work.

The time scheduling system has been developed with the object of appreciably lowering the cost due to downtime (caused by breakdown or failure of machinery or equipment) of industrial installations. It is based on the principle of grouping maintenance operations together as much as possible (Fig. 4).

(3) Structure of a reporting and information system:

Thanks to a new approach to the storage of master data (Fig. 5), a sound basis for the detailed reporting and the dialogue oriented information system is obtained. The storage technology envisaged in Fig. 5, embodying the modular principle, is used also for linking the maintenance operations with the units of plant.
For example, with this system, in the event of an unscheduled shoutdown, appropriate maintenance jobs can immediately be suggested which can be properly carried out within this shutdown period.

(4) What the system comprises:

In its standard form the WARTAS system comprises the following:

- A tested program package in a high level programming language, e.g., COBOL.

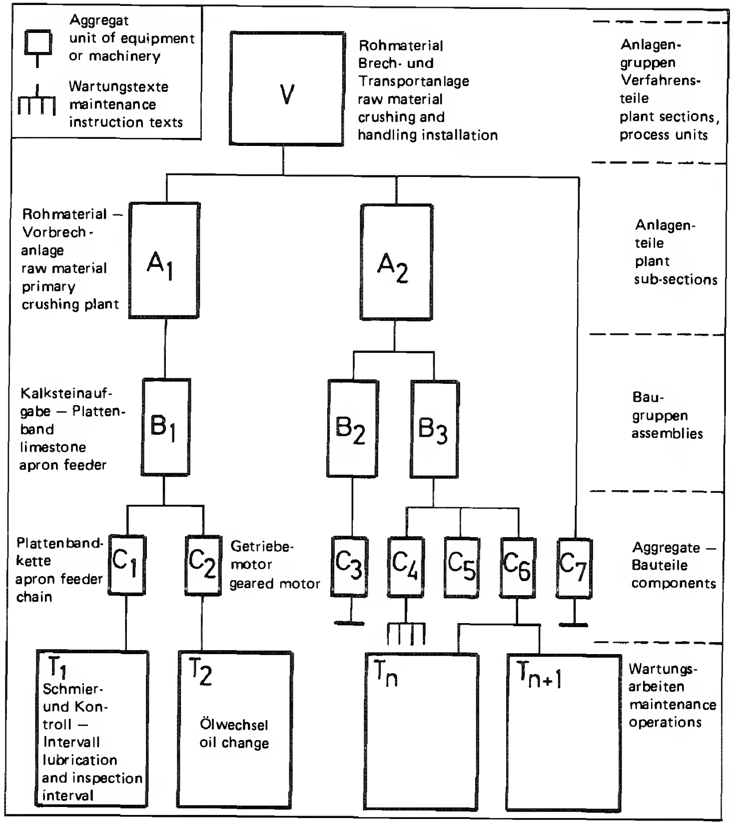

Fig. 4: Example of the storage of data for part of a plant, with maintenance operations to be carried out (KHD Humboldt Wedag AG)

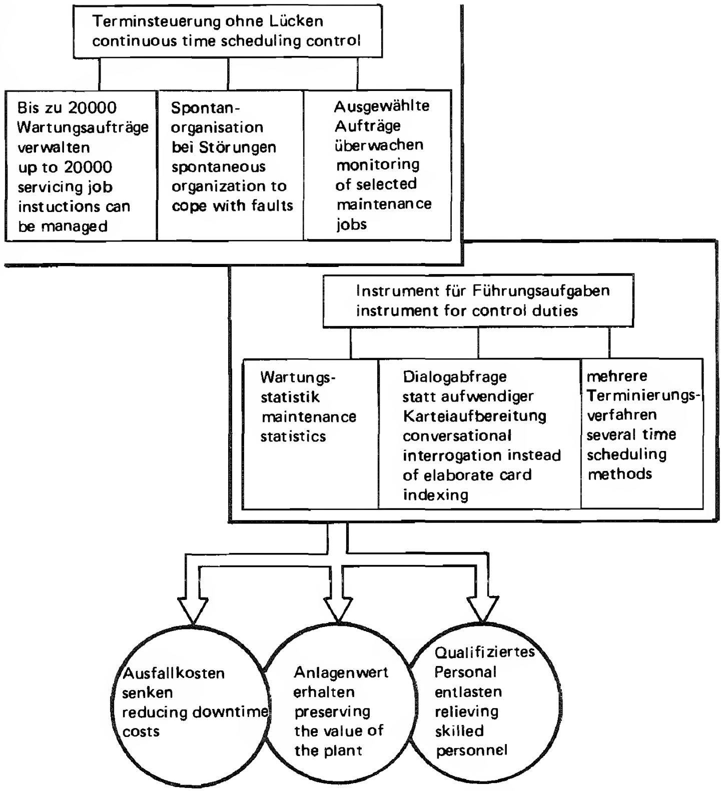

Fig. 5: Time scheduling targets (KHD Humboldt Wedag AG)

– A computer suitable for conversational operation (dialogue), with video units, discs, input/output cards, high-speed printer, and data safeguarding tape. The disc storage capacity is at least 20 million bytes and the working storage capacity at least 64 million bytes.
– On-the-spot commissioning, training of customer's personnel, documentation. The customer is moreover given the option of a package deal offering "turnkey" ready-to-use master data prepared by specialist engineers in cement manufacturing technology and plant maintenance.

The following advantages can be claimed for computerized maintenance control as provided by the system outlined here:

- The cost arising from unplanned shutdown is reduced thanks to a time scheduling system specially designed to cut downtime.
- The value of the plant is preserved and protected by full information on the required maintenance work, issued in good time.
- Skilled personnel are relieved of routine duties.

The combination of personal and machine-based organization enhances the responsiveness of the maintenance system to any contingencies that may arise. The more evenly spread load makes for higher capacity and effectiveness of manpower in this sphere of activities which still offers such scope for rationalization.

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II. Problems of wear

The section "Maintenance" cannot be considered in isolation from the problems of wear in cement works engineering. These problems belong to the science of tribology – a comprehensive term denoting the study of friction, wear and lubrication, and in general the behaviour of interacting surfaces in relative motion.
Wear in engineering components is caused by friction, impact, percussion, heat and/or chemical action. There is hardly any section in a cement works where some kind of wear does not occur. The large dimensions of the machinery and the high rates of material throughput in all stages of the process are associated with considerable wear of the parts involved, causing the progressive loss of a variety of materials which therefore have to be replaced from time to time: many different grades of steel, refractories, other ceramic materials, rubber, etc.
This being so, cement works operators have, in collaboration with the manufacturers of their machinery and with the suppliers of the materials involved, made every effort to bring wear under control. As a result, considerable success has been achieved, as the following example will serve to show:

About twenty years ago the specific rate of grinding media wear in finishing mills (clinker grinding) was in the range of 800 to 1000 g per t of cement, whereas the corresponding figure is now only about 50 to 120 g/t.
The problems associated with wear are obviously very varied and sometimes quite complex. To attempt anything like a detailed treatment of subject would therefore be outside the scope of this book. The reader wishing to obtain further information should consult the relevant literature, a selection of which is listed in the appended bibliography.

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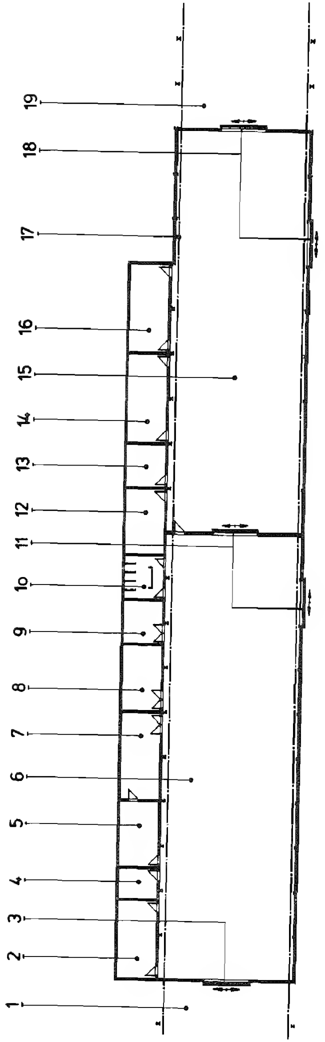
K. Workshops and spare parts store

By B. Kohlhaas

Every cement works should be equipped with facilities for carrying out routine repairs in its own workshops. The advantages are, among others, that the personnel entrusted with this work are familiar with the machines and appliances requiring their services and that often there is a substantial saving in time and cost if repairs can be executed "on the premises".
Obviously, this requires the availability of efficient workshops with equipment properly suited to the needs of the works. This is especially important in that cement works are often located far from other industrial plants, so that it may not be conveniently possible to get outside help.
In this section an outline proposal for the equipment of the workshops for a fairly large cement works is presented. Of course, depending on local factors and circumstances, changes can be made in the range of equipment and in the size or capacity of the recommended machine tools.
A store containing the spare parts in regular demand must back up the workshop facilities. Large and heavy parts such as, for example, kiln tyres, supporting rollers, girth gear sections, etc., which not only take up a considerable amount of storage space but also require powerful lifting and handling appliances for their manipulation, should be stored as close as possible to the actual machinery to which they will have to be fitted.
The accompanying drawing shows a suggested layout for workshops and ancillary facilities, together with a spare parts store. Good accessibility and suitable equipment, with adequate lifting appliances, are essential to effective utilization of these installations.
The workshop and ancillary equipment will be considered under the ten following headings:

1.00 Machine shop
2.00 Fitters' shop
3.00 Welders' shop
4.00 Smiths' shop
5.00 Electrical repair shop
6.00 Erection equipment
7.00 Joiners' shop
8.00 Toolmakers' shop
9.00 Painters' shop
10.00 Central compressed air supply unit.



Workshops and spare parts store

1 erection area and extension
2 job planning and costing
3 sliding door
4 foreman's office
5 tool crib
6 workshop
7 smiths' shop
8 fitters' shop
9 electrical repair shop
10 WC and wash room
11 sliding door
12 materials issue and stores inventory card file
13 stores manager
14 store for small electrical components
15 spare parts store
16 motor store room
17 crane track
18 sliding door
19 storage area and extension

1.00 Machine shop

1.01	1	power hacksaw for round material up to 200 mm diameter and square material up to 200 mm × 200 mm	
1.02	1	universal shearing machine for cutting, punching and perforating of flat and sectional material:	
		flat steel bars up to	150 mm × 16 mm
		plates up to	13 mm
		round steel bars up to	42 mm
		square steel bars up to	38 mm
		perforates up o 25 mm diameter in 16 mm material at approx. 50 t pressure rating and 27 mm stroke	
1.03	1	engine lathe/sliding lathe	
		height of centres	250 mm
		distance between centres	2000 mm
1.04	1	ditto with gap bed	
		maximum machining diameter over bed	660 mm
		largest machining diameter over gap	860 mm
		distance between centres	4000 mm
1.05	1	radial drilling machine	
		for drilling in steel up to	50 mm
		for drilling in cast iron up to	63 mm
1.06	1	transportable universal radial drilling machine	
		for drilling in steel up to	50 mm
1.07	1	upright drilling machine	
		for drilling capacity (diameter drilled) up to	32 mm
		drilling depth up to	200 mm
1.08	1	shaping machine for maximum machinable length	710 mm
1.09	1	universal column milling machine	
		work mounting area	450 mm × 2000 mm
1.10	1	single-spindle threading machine for metric die heads	10 – 52 mm
		B.S. pipe thread and Whitworth thread	$^1/_2$" – 4"
1.11	1	hydraulic single-column straightening press	
		pressure rating	100 t

2.00 Fitters' shop

2.01	1	double wheel stand for	
		grinding wheel diameter	150 mm
		grinding wheel width	20 mm
2.02	2	double wheel stands for	
		grinding wheel diameter	300 mm
		grinding wheel width	40 mm
2.03	4	work benches	
2.04	2	assorted vices	
		pipe vices for lugs up to 3"	
		opening	90 mm
	12	parallel-jaw vices	
		jaw width	125 mm
		opening	175 mm
		depth	145 mm
2.05	2	drilling frames with pneumatic drills	
		drilling capacity (diameter drilled)	32 and 50 mm
		height	640 and 770 mm
		travel	240 and 305 mm
		max. feed pressure	800 and 1100 kg
2.06	1	hand lever shearing machine for	
		flat bars up to	60 mm × 5 mm
		round bars up to	13 mm dia.
2.07	1	mandrel press with pillar spindle with flywheel	

3.00 Welders' shop

3.01	2	single-housing arc welding machines on travelling carriage, connected load 15 kW at 100% duty cycle	
		control range	40–600 amps
		open-circuit voltage	60–100 volts
		with remote control, connecting cable, welding cable, electrode holder, etc.	
3.02	1	oxygen cutting machine	
		cutting range	3–100 mm
3.03	1	CO_2 arc automatic welding machine	
		max. welding current at 100% duty cycle	400 A
		welding voltage	15–34 V
		rated power	17.5 kW
		cooling water pumping system	
		spot welding and tack welding machine	
		welding guns, CO_2 preheaters	

3.04	1	welding edge preparation machine	
3.05	3	or more complete sets of electric welders' equipment, including 2 welding work tables (approx. 1000 mm × 800 mm × 800 mm)	
3.06		sets of equipment for several oxyacetylene welders, including high-pressure acetylene gas generators, welding and cutting torch sets, pressure reducing valves, gas and oxygen hoses, cylinder trolley	

4.0 Smiths' shop

4.01	1	double-hearth forge fire plate dimensions 2000 mm × 1250 mm, including integral electric blower, air intake pipeline, draught damper, chimney	
4.02	1	anvil, 200 kg, with base, horn, forging tools and 1 set of swages 10–25 mm	
4.03	1	swage block 500 mm × 500 mm	
	1	straightening block 1500 mm × 1000 mm × 40 mm high	
	1	blacksmith's vice	
4.04	1	powered spring hammer weight of head approx. 60 kg, for the forging of 90 mm square, 90 mm dia. round, and 50 mm × 160 mm flat bars	

5.0 Electrical repair shop

5.01	1	engine lathe/sliding lathe	
		height of centres	800 mm
		distance between centres	
5.02	1	light bench drilling machine	
		drilling capacity (dia.)	6 mm
		drilling depth	60 mm
5.03	4	electric two-speed drilling and percussion drilling machines	
		speeds	2
		drilling capacity in steel (dia.)	10/6 mm
5.04	2	angle drilling machines	
		drilling capacity in steel (dia.)	6 mm
5.05	1	lever shearing machine for	
		plates up to	5 mm
		flat bars up to	60 mm × 6 mm
		round bars up to	13 mm dia.
5.06	3	steel work-benches, dimensions	1500 mm × 700 mm

5.07 various measuring instruments:
2 Wheatstone bridges for the measurement of ohmic resistances
1 multi-range instrument for
0.03 – 600 mA and up to 1.5 A
0.45 – 600 V for d.c. 0.3 – 300 mA and 1.5 – 6 A
6 – 600 V for a.c.
for resistances 0 – 10 K ohm
0 – 10 M ohm
2 straight-through current transformers for measuring ranges 0 – 150 A and 600 A
1 slide-wire bridge for
0.04 ohm – 6.4 megohm in eight ranges and two voltage ranges 10 V and 100 V
1 insulation tester 0 – 500 megohm and 0 – 1000 megohm
2 clip-on wattmeters with
four measuring ranges 10 – 50 – 250 – 500 A for a.c. and two measuring ranges 250 – 500 V for a.c.
1 recording multi-range instrument

for d.c.	0 – 600 mA
	0.012 – 600 V
for a.c.	3 – 600 mA
	6 – 600 V

including carrying case
1 voltage tester with case
8 continuity testers 30 ohm
2 wattmeters for connection in series 150 kW

5.08 tools for electricians
2 small sets of tools (for electricians on shift work)
general tools (it is advisable to provide several complete tool kits)

6.00 Erection equipment

6.01 1 electric hoist for 5000 kg lifting capacity rope speed 11 – 18 m/min, rope length 20 m

6.02 various items of erection/assembly equipment
2 pulley blocks 10 t
4 pulley blocks 7.5 t
4 pulley blocks 5 t
16 snatch blocks
6 ratchet hoists 4.5 t, 1.5 m lifting height
6 ratchet hoists 6.0 t, 1.5 m lifting height
6 lifting-clamp hoists 1.5 t
6 lifting-clamp hoists 3.0 t
2 chain hoists 10.0 t, 6.0 m lifting height

2 jacks 200 t
4 jacks 100 t
4 jacks 50 t
2 force pumps (water), effective capacity 132
1 force pump (water), effective capacity 5.52
1 force pump (water), effective capacity 2.52
various pipelines and fittings
100 m of each of the following wire rope diameters: 14 mm, 17 mm, 19 mm
1 welding and lighting set, 45 kVA, portable

7.00 Joiners' shop

7.01 1 band saw
1 mortising machine
1 circular saw
1 planing machine
column drilling machine
1 pendulum saw
1 band saw butt-welding machine
1 saw sharpening machine
1 planer knife sharpening machine

7.02 various hand tools

7.10 plumbers' tools: 3 complete tool kits

7.20 pipe fitters' tools: 4 complete tool kits for gas pipe fitting, threaded pipe fitting, lead pipe fitting, copper pipe fitting, with 1 set of welding and cutting torches and 1 cylinder trolley

8.00 Toolmakers' shop

8.01 1 marking-off and surface plate 2500 mm × 1250 mm
1 pair of tailstocks with spring spindles
1 pair of parallel blocks 90 mm × 200 mm × 170 mm
2 pairs of parallel blocks 60 mm × 30 mm × 260 mm
2 pairs of parallel blocks 80 mm × 40 mm × 320 mm
1 pair of prisms 250 mm long
1 scribing block, 750 mm scribing height
1 measuring stand with adjustable column, 750 mm height
1 parallel box 300 mm × 150 mm × 500 mm
1 marking-off and angle plate,
horizontal surface/height 300 mm × 180 mm × 630 mm
1 measuring jig for 40 mm × 8 mm × 1000 mm bar gauge
1 bar gauge 40 mm × 8 mm × 1000 mm

	1	case of slip gauges, 46 pieces
	1	case of auxiliary tools for slip gauges, range 0–200 mm
	2	cabinets for storage of tools
8.02	1	universal tool grinding machine
8.03	1	electric double wheel stand with grinding attachment for twist drills
8.04		general equipment for a tool crib, with the requisite measuring tools, tools for all the tradesmen employed in the cement works, lifting and handling appliances, hand drilling machines, hand grinding machines, etc.

9.00 Painters' shop

9.01	3	complete sets of tools such as stopping knives, scrapers, putty knives, glass cutters, abrasive blocks, etc.
9.02	3	complete sets of paperhangers' equipment such as pasting tables, rollers, etc.
9.03	3	complete sets of brushes of various types and shapes
9.04	3	sets of flat brushes, paint rollers
9.05	3	paint spray guns including 3 breathing masks, blow-out guns
9.06		various receptacles, buckets, etc.
9.10	1	portable compressor, single-stage up to 8.8 bar max.

10.00 Central compressed air supply unit

for supplying the various workshops with compressed air for driving and cleaning purposes, equipped with a compressor delivering about 11 m^3/min at an operating pressure of 7.0 bar, including cooler, air receiver, piping, electric drive and switchgear for automatic control.

L. Water supply, compressed air

By B. Kohlhaas

I. Water supply for cement works

1 Estimated quantities required

The water demand of a cement works depends very much on the production method and the machinery used. It also depends, though to a less extent, on the actual size of the works. The main consumer items are the production installations with their ancillaries, the laboratory, the personnel facilities, offices and residential accommodation.

The water supply arrangements should be qualitatively and quantitatively suited to the requirements. In many countries there are statutory regulations to be satisfied as regards fresh water supply and waste water disposal. The capital and operating costs to be borne by the cement works operator can be very substantial. Careful planning is therefore essential and should cover all aspects: supply, preparation and waste discharge for the production, administrative and residential sectors. For major cement works developments and/or in difficult water supply situations it is advisable to enlist the aid of a specialized consultants' firm in order to achieve optimum technical and economic optimization. In situations where water is in short supply it is usually necessary to operate with closed-circuit systems as far as possible.

For planning a water supply system the first step will normally consist in estimating the quantity of water needed. It comprises:

(a) service water, i. e., non-potable water for industrial use, more particularly for cooling, which is recooled and re-used over and over again;
(b) drinking water (potable water);
(c) water which is lost by evaporation or leakage and therefore has to be replaced.

Depending on local conditions (e. g., climate) and the production method, the quantity of water required will vary. For the dry process of cement manufacture,

now predominantly used, the following approximate values provide guidance for Central European climatic conditions:

Table 1: Approximate values for water requirements

(1) Service water (cooling water)

plant output	t clinker/day	800	1000	2000	3000
cooling water consumption	approx. m^3/h	60	80	100	120
10% for leakage loss	approx. m^3/h	6	8	10	12
sprayed water (lost by evap.)	approx. m^3/h	8	10	20	28
losses to be made good with make-up water	approx. m^3/h	14	18	30	40

(2) Drinking water

Drinking water consumption can be put at about 100 m^3/day (corresponding to roughly 0.5 m^3 per employee), with possible short-term peak demand of 20–30 m^3/hour on account of shift working. It is therefore advisable to have a stored supply of at least 25 m^3, e.g., in a water tower.
A drinking water supply rate of not less than 5 m^3/hour is recommended.

(3) Total water consumption (fresh intake from the supply)

plant output	t clinker/day	800	1000	2000	3000
service water	approx. m^3/h	14	18	30	40
drinking water	approx. m^3/h	5	5	5	5
total water	approx. m^3/h	19	23	35	45
daily consumption	approx. m^3/d	480	652	840	1080

During the periodic heating and cooling of the water in the closed-circuit cooling system a proportion is lost as a result of evaporation in the recooling plant (cooling tower, spray pond) and of leakage. Water sprayed into grinding mills, electrostatic precipitators, etc. is also lost, as is the water used for cleaning purposes or consumed as drinking water.
Additional water must therefore constantly be fed into the system to make up for losses and to ensure that the circulating water for cooling will not undergo changes in its chemical composition or physical properties.

In tropical, subtropical or very dry climates the quantities needed may differ considerably from the guide values given in the above table.

2 Raw water

2.1 Condition of raw water

No standard requirements have yet been laid down for the condition of the water used in the installations of cement manufacturing works. On the basis of general experience, however, the water circulating in the cooling system should have the following properties.

- it should be of such temperature that effective heat exchange in the installations to be cooled is obtained;
- it must not have an aggressive or corrosive effect on metal and concrete parts of the system;
- it must not promote or sustain the growth of organisms in the system;
- it must not contain oil or grease.

Furthermore, it should conform to the following limiting values:

designation	unit	limiting values
pH value	–	7–9
carbonate hardness	°n	4–14
	mval/l	1.4–5.0
total salt content	mg/l	3000
chlorides	mg/l	500
sulphates	mg/l	500
total content of chlorides and sulphates	mg/l	500
iron	mg/l	1.0
manganese	mg/l	0.15
magnesium	mg/l	60
suspended solids approx. 0.05 mm	mg/l	40
temperature on entry to circuit, max.	K	303
	(°C)	(30)
average rise in temperature	K	298–303
	(°C)	(25–30)

A further important criterion for judging the suitability of water is its stability on Heating. Water which forms no calcium carbonate deposits on being heated to 313–323 K (40°–50° C) and cooled to 292–303 K (20°–30° C) is to be rated as stable. If it fails to meet this requirement, it will have to be treated.

2.2 Water winning

For obtaining the necessary supply of water, any sources existing in the neighbourhood of the cement works – lake, river or stream – should first be investigated. Wells drilled on or near the works site may provide an additional supply. Only if such sources are unavailable or inadequate to meet the needs should water from the public supply system be used.

2.3 Preparation of the water

As mentioned above, the properties of the raw water have to satisfy certain chemical and physical conditions. Compliance with these requirements should be ensured in the interests of health and for technical reasons of trouble-free plant operation.
It is therefore necessary to examine and analyse the water before building a new cement works. Subsequent routine tests to check the condition of the water during operation of the plant should also be carried out. If the quality of the water satisfies the conditions, no special preparatory treatment need be provided. However, if treatment is needed, it may – in order to keep down the cost – be limited to the following:

- reduction of the quantity of suspended solids;
- suppression of the tendency to form calcium carbonate deposits (stabilization);
- reduction of corrosive properties;
- prevention of the growth of organisms in the water system.

The suspended solids quantity can be reduced by sedimentation in settling basins. This process can be accelerated by the addition of coagulants such as aluminium sulphate $Al_2(SO_4)_3$ in conjunction with CaO or Na_2CO_3.
Various inorganic compounds – e. g., sodium phosphate, trisodium polyphosphate or silicates – can be used for stabilizing the cooling water and at the same time reducing its corrosiveness. The substance most widely employed in recent years is sodium silicate.
The growth of undesirable organisms in the system can be suppressed by chlorination (chlorine gas) or the addition of sodium hypochlorite.
If the cement works water supply system is required also to provide drinking water, more elaborate treatment may be necessary for that purpose. More particularly, the water will have to be subjected to appropriate bacteriological purification.

3 Supply system, cooling water circuit, water storage

The pumps for delivering the water to the works and those for the in-plant water supply and distribution system should have adequate reserve capacity in order to minimize the risk of breakdown in supply or in cooling water circulation if a technical fault occurs. The intermediate storage tanks for raw water, treated water

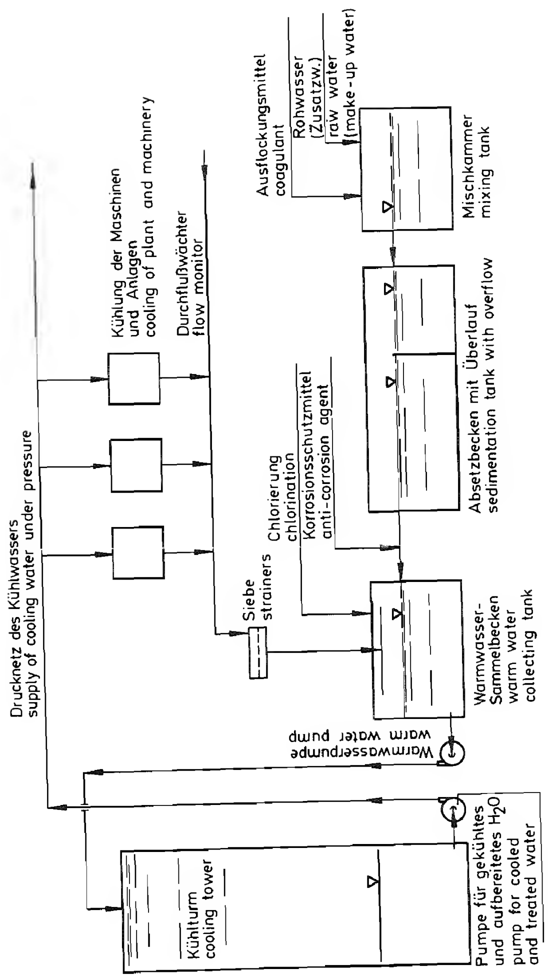


Fig. 1: Diagram of a cooling water system (not including drinking water purification)

(drinking water and service water respectively) and the warm water collecting basin should be amply dimensioned.
Fig. 1 schematically shows a cooling water circuit with preparation of the make-up water. Instead of the settling basin a gravel filter with backwash system may be used. A large basin or pond with spray equipment may be substituted for the relatively expensive cooling tower.

4 Waste water disposal

In many countries the discharge of sewage and other effluents from industrial installations, laboratories, residential premises, etc. is subject to statutory regulations, which will duly have to be considered in connection with the design of water supply systems.

II. Compressed air supply

A compressed air supply system is a great asset in a cement works. Compressed air is used for a wide range of purposes: for example, it is used for driving pneumatic drills and other tools for repair and maintenance work, for cooling certain measuring instruments, for cleaning parts of machinery such as bearings, gears, motors, etc.
A very advantageous arrangement is to provide a central compressor installation connected to the various consumer sections of the works by a ring main. For convenience of supervision these compressors should be located in immediate proximity to the workshops, where a separate compressor installation for pneumatic equipment in workshop use is often provided anyway.
Of course, it is alternatively possible to adopt a decentralized compressed air supply system by installing compressors at various strategic points in the cement works, but it is a more expensive alternative in terms of capital cost.
A third solution for the compressed air supply consists in providing a central installation, as described above, but connected to a radiating system of pipes to the individual consumer sections of the works.
For example, the following equipment is recommended for a central compressor installation in a cement works with a capacity of 3000 t/day (see Figs. 2, 3 and 4).

3 compressors, each with an effective delivery capacity of about 11 m^3/minute at normal operating pressure of 7.0 bar or

2 compressors, each with an effective delivery capacity of about 16 m^3/minute at normal operating pressure of 7.0 bar
2 recoolers, each for cooling the air at a rate of 11 m^3/minute, water requirement about 0.9 m^3/hour, cooling water entry temperature 10° C
1 compressed air receiver, 8000 litres capacity operating pressure 10 bar
test pressure 13 bar
operating temperature 120° C
with attached compressed air distributor system
1 switch device for controlling the compressors
1 set of various pipelines in the compressor installation, including fittings.

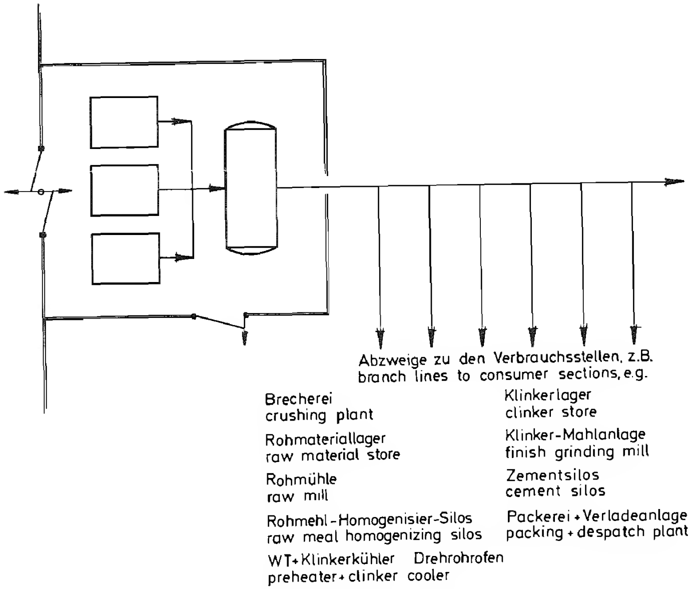


Fig. 2: Central compressor installation with ring main supply system to consumer sections

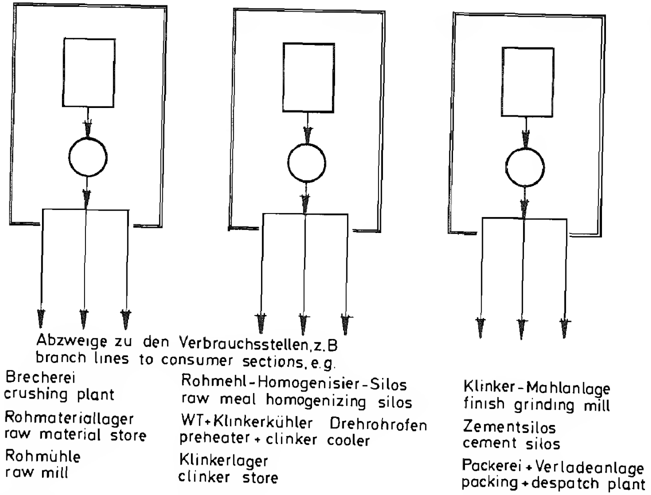


Fig. 3: Central compressor installation with radial supply system to consumer sections

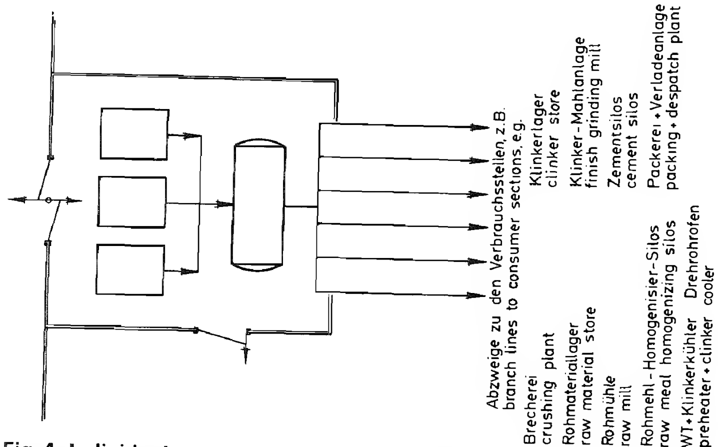


Fig. 4: Individual compressor installations located near main consumer sections in various parts of the cement works

M. Personnel requirements

By B. Kohlhaas

The number and type of personnel required by a modern cement works will depend on geographical location and local conditions. Other determining factors are the type of machinery in the works and the degree of automation of the manufacturing process. The method of manufacture (dry or wet process), on the other hand, is of little influence on plant manning levels required. Major differences may occur, however, in the case of cement works in tropical or subtropical countries, where there is often an abundance of available labour. In such countries the level of productivity per employee tends to be lower than in those with temperate climates where natural conditions are more favourable in this respect and wage rates are usually also higher.
The data given below are based on a cement works producing 1000 tonnes per day, using the dry process and assumed to be equipped with the most up-to-date technology. They can, of course, be no more than very approximate guiding values and will have to be scrutinized and, if necessary, corrected to take account of local conditions.

Table 1: Cement works personnel

A. Salaried staff

(1) General administration

1 general manager in charge
1 secretary
1 office manager
2 clerical assistants
1 works doctor
1 medical assistant
1 messenger

(2) Technical administration

1 technical manager in charge
1 secretary
1 production manager (possibly, chief chemist)
1 laboratory manager (chemist)
1 quarry manager (geologist)
4 shift engineers (mechanical or electrical)

1 engineer for works maintenance and safety
1 electrical engineer
4 shift foremen
1 mechanical workshop foreman
1 electrical workshop foreman
2 messengers

(3) Commercial administration

1 commercial manager in charge
1 secretary
1 marketing manager
4–6 marketing assistants
1 purchasing manager
2–3 purchasing assistants
1 stores administrator
5 accounts clerks
2 wages and salaries clerks
1 pensions clerk
5–10 typists (possibly in a typing pool)
2 messengers

B. Wage earners (skilled and semi-skilled)

(1) Laboratory

6 laboratory technicians
3 helpers

(2) Quarry

1 ganger (who may also be the shotfirer for blasting)
3 drilling machine operators (1 on stand-by)
3 excavator operators (1 on stand-by)
1 loading shovel operator
4 truck drivers (1 on stand-by)
10 helpers

(3) Limestone crushing and clay preparation plant

1 crusher operator
1 attendant for clay preparation
2 helpers

(4) Raw materials store

3 storeyard attendants

(5) Raw material silos

4 silo attendants

(6) Raw grinding and finish grinding plants

4 mill attendants
helpers

(7) Rotary kiln plant with preheater, clinker cooler and electrostatic precipitator

4 burners (1 on stand-by)
4 preheater and cooler attendants
3 attendants for electrostatic precipitator
3 greasers

(8) Clinker and gypsum stores, store for admixtures (if any)

3 store attendants
3 helpers

(9) Cement silo installation and packing/despatch plant

2–3 packing machine operators
2 helpers
3 loaders
2 silo attendants
2 helpers in sack store

(10) Mechanical workshop

6 fitters
6 shift mechanics
6 helpers

(11) Electrical workshop

4 electricians or electronics technicians
4 shift electricians
4 helpers

(12) Building maintenance

1 head bricklayer
2 bricklayers
4 helpers

(13) Spare parts store

2 storemen
2 general helpers

(14) Auxiliary facilities

(a) Oil, gas and water supply*)
2 attendants
2 helpers

(b) Safety and security services
4 doorkeeper-checkers
9 watchmen
3 firemen (trained in fire prevention and firefighting)
3 first-aid men (trained)

(c) General duties
6 crane drivers
3 messengers

*) Note: If solid fuel (e.g., coal) is used instead of oil or natural gas, the following additional operatives will be needed
1 attendant for coal stockpile and pulverized coal silo
3 coal mill attendants
2 helpers

N. Lubricants, storage and consumption

By B. Kohlhaas

I. General

Efficient lubrication with lubricants best suited for the particular purpose is essential to maintaining operational reliability of machinery and prolonging its useful life. The lubricants for gears, plain and anti-friction bearings, etc. should therefore be procured and applied in consultation with the suppliers of these lubricants and in accordance with the suggestions of the machinery manufacturers.

In general, it is important to comply with these recommendations:

- always use the right lubricant for any given purpose;
- make sure that lubricants are correctly stored;
- keep a careful check on lubricant consumption.

II. Types of lubricant

The following main categories are to be distinguished:

(1) Oils

- hydraulic, bearing and circulating oil
- anti-freezing bearing and circulating oil
- gear oil
- bearing and circulating oil for very high temperatures (cylinder oil)
- running-in oil for gears
- insulating and transformer oil

(2) Greases

- water-repellent grease
- water-resistant grease
- gear grease
- high-temperature bearing grease

(3) Special lubricants

- bituminous lubricant containing solvent
- spray-adhesive lubricants

The principal properties of the various lubricants are listed in Table 1.

III. Storage of lubricants

1 Delivery and handling

Lubricants in relatively large quantities are for the most part supplied in metal drums, while smaller quantities are packed in tins or cans. As a rule, these containers are delivered to the consumers by rail or road vehicle. Negligence in handling them in transit or on unloading at their destination is liable to damage the containers, resulting in losses. It is therefore important to exercise proper care. Suitable handling appliances for unloading from the delivery vehicles will facilitate dealing with heavy drums, protect them from damage and help to prevent accidents.
The normal drum weighs between 180 and 220 kg. It should therefore certainly not be unloaded by dropping it from the transporting vehicle – not even if it is allowed to fall on a cushion of old motor tyres or something of that kind. The impact of the fall is liable to cause the seams of the drum to open; there is moreover a considerable accident hazard when such crude methods are applied.
For transporting the drums within the works, suitable vehicles may be employed, such as electric trucks, platform trucks, drum transporter trucks, etc. In many

Table 1: Principal properties of lubricants (from KHD Humbold Wedag AG)

Identi-fication No.	Type of lubricant	ISO-VG viscosity class	Kinematic viscosity mm^2/s $\pm 10\%$	Flash point COC min. °C	Pourpoint min °C	FZG load stage
OL 1	Hydraulic, bearing and circulating oil	46	46	220	−24	12
OL 2	Anti-freezing bearing and circulating oil	46	46	190	−36	7
OL 3	Gear oil containing mildacting E.P additives	68	68	210	−24	12
OL 4	Gear oil containing mildacting E.P. additives, bearing and circulating oil	100	100	220	−21	12
OL 5	Gear oil containing mildacting E.P. additives	220	220	220	−18	12
OL 6	Gear oil containing mildacting E.P. additives, bearing and circulating oil	320	320	230	−18	12

Identi-fication No.	Type of lubricant	ISO-VG viscosity class	Kinematic viscosity mm^2/s $\pm 10\%$	Flash point COC min. °C	Pourpoint min. °C	FZG load stage
OL 7	Gear oil containing mildacting E.P. additives, bearing and circulating oil	460	460	230	−15	12
OL 8	Bearing and circulating oil for very high temperatures (Cylinder oil)	1000	1000	280	− 6	8
OL 9	Insulating and transformer oil	–	8−10	130	−51	–
OL 10	Running-in oil for gears	220	220	220	−18	12

Identi-fication	Type of lubricant	Symbol according to DIN 51 502	Penetration worked at 25°C 1/10 mm	Drop point min. °C	Temperature range °C
F 1	High-grade, water-repellent calcium base grease	M 2b	265−295	100	−20− + 50
F 2	Water resistent, soft Lithium base grease	K-L 2k	265−295	185	−30− +130
F 3	Water resistent lithium base grease of medium consistency	K-L 3k	220−250	185	−30− +130
F 4	Fibrous sodium base gear grease containing E.P.-additives	G-00f	400−430	150	−15− + 80
F 5	High temperature bearing grease	H 20	265−295	260	−15− +150

Identi-fication No.	Type of lubricant		Mode of lubrication
H 1	Bituminous lubricant with solvent content for use in open gears, chains and ropes		By hand
H 2	Special lubricant for lubrication of gears exposed to high external temperature (100°C)	See special Lubrication Instructions	a) Autom. spray lubrication b) Circulation oiling c) Splash lubrication d) By hand

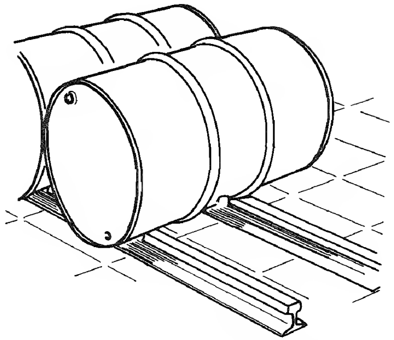


Fig. 1: Transporting drums in the works by **rolling on rails**

instances, however, the drums are simply rolled along. In that case suitable rail tracks should be provided, if possible, as these will help to prevent damage and dirtying of the drums and will make their transport easier (Fig. 1).
Large consignments of mineral oil are delivered in bulk, generally in rail tank waggons, sometimes also in road tank vehicles or in demountable tanks carried on vehicles. Emptying can be a fairly simple operation if the waggons or vehicles can be brought close enough to the storage tanks or to the lubricating system itself, so that the oil can flow out by gravity. If the distances to be overcome are longer, or if the tanks to be filled are mounted at a higher level, a pump connected to them by hoses will have to be used. If the oil has to be pumped over greater distances, it will be advantageous to install a permanent pipeline ($1^1/_2$ or 2 inch bore).

2 Storage

The storage facilities should be of sufficient capacity to suit the cement works' needs. For guidance, the following figures represent approximately one year's supply for a works with an output of 3000 t/day·
110000 litres of oil in various grades, in about 690 drums; 32000 kg of grease in various grades, in about 280 drums.

2.1 Outdoor storage

As a general rule, oil and grease should be stored in enclosed buildings or rooms kept at a fairly even temperature. Wind and rain should be excluded if only because they are liable to obliterate the markings and designations on the drums. Besides, moisture or frost may cause deterioration of quality
However, if intermediate storage in the open air cannot be avoided, the drums should at least be kept under the protection of a roof structure (Fig. 2)

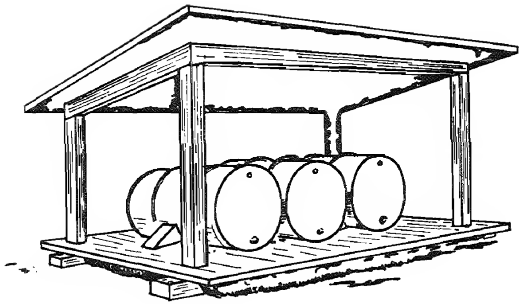


Fig. 2: Outdoor storage of drums

If even this minimum protection is unavailable, it must be ensured that the drums are stored in the reclining position, i. e., not upright. It is advisable to place them on rails or on timbers and to cover them with tarpaulins or roofing felt.
If drums are left standing upright and unprotected, there is a risk that water and dirt will enter through a plug that has not been properly closed. Not so well known is the fact that substantial amounts of water can get into a drum even through a properly closed plug under open-air storage conditions. This is due to the suction effect associated with the "breathing" of the drum caused by temperature variations. See Fig. 3.
Such ingress of water, which may amount to several litres in a few days, can render the oil useless, particularly in the case of special oils such as, for example transformer oil or oil for very low-temperature duty.

2.2 Storage in buildings

2.2.1 Lubricants store

A room intended for the storage of lubricants should be dry and have as even a temperature as possible or at least be frost-free.

2.2.2 Location of store

The location of the lubricants store should primarily depend on the disposition of the other buildings, the major consuming machinery, and the in-plant transport routes. Good access for vehicles delivering the lubricants and adequate handling facilities (unloading platform, lifting appliances) are essential to quick and reliable reception of these materials on delivery. A railway connection is necessary if large quantities are involved.

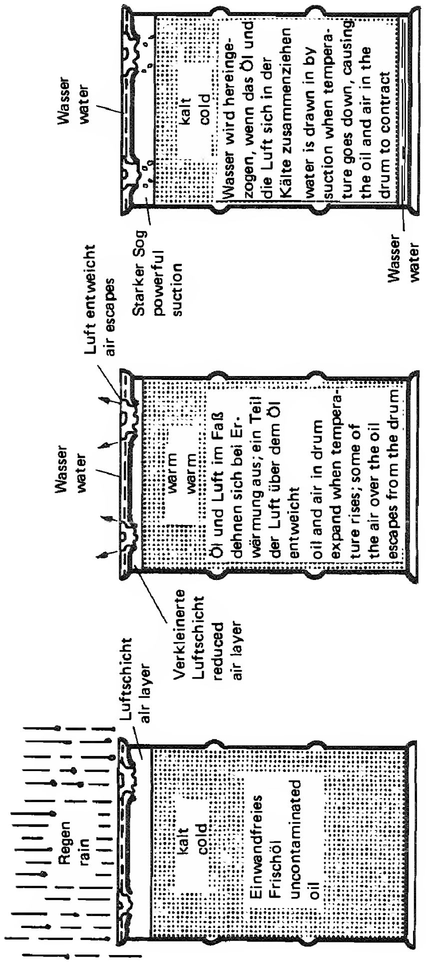


Fig. 3: How "breathing" draws water into an oil drum

2.2.3 Size of store

The size – i. e., the capacity – of the store will depend on the number of different grades or types of lubricant to be stored, and the maximum quantities of each. Planned future works extensions, if any, should also be allowed for. As a general rule, it is advisable to base the storage capacity on about two to three months' supply of lubricants.

2.2.4 Construction

Lubricants are in general not easily ignitable substances, so that their storage arrangements do not have to meet any special requirements as to fire safety. However, cleaning fluids and solvents are often also stored in the same room or building as that used for lubricants, so that the statutory regulations relating to combustible liquids will then have to be complied with. Under the relevant German regulations the following classification applies to liquids which are immiscible (or only partly miscible) with water (category A):

- hazard class I. flash point below 21° C;
- hazard class II. flash point of 21° to 55° C;
- hazard class III: flash point of 55° to 100° C
  (flash point determined in closed crucible by Pensky-Martens method).

Store rooms for such liquids should conform to special safety requirements, and if lubricants are stored together with these flammable substances, the safety regulations apply to the entire storage space.
For example, precautions must be taken to ensure that any leaked liquids cannot make their way into adjacent rooms. Doors should therefore have sufficiently high thresholds, provided with a ramp on each side to enable trucks to enter and leave the store room. Doors should be of fire-retardant steel construction and should open outwards. Windows should be fitted with wire glass. All electrical apparatus should be flame-proof (or explosion-proof) enclosed, in accordance with the relevant regulations. An adequate number of hand fire extinguishers should be available, likewise in accordance with regulations (for example, German safety codes require a at least six extinguishers for "fire class B, fire hazard rating b").
In view of these requirements it is generally advantageous to provide a separate store room for flammable liquids, as exemplified in Fig. 4.
The statutory requirements for the prevention of ground-water pollution must also be complied with. More particularly, it must be ensured that no oil that may escape from drums or tanks can seep into the ground. No oil is allowed to be discharged into the public sewerage system either. In some countries, as in the Federal Republic of Germany, the interpretation and enforcement of the pollution control laws is entrusted to local government authorities, who should therefore be consulted in connection with the planning of storage facilities.
The lubricants store should have effective ventilation. Good lighting is also important because it helps to prevent mistakes due to misreading of labels or instructions and also promotes general cleanliness in the store.

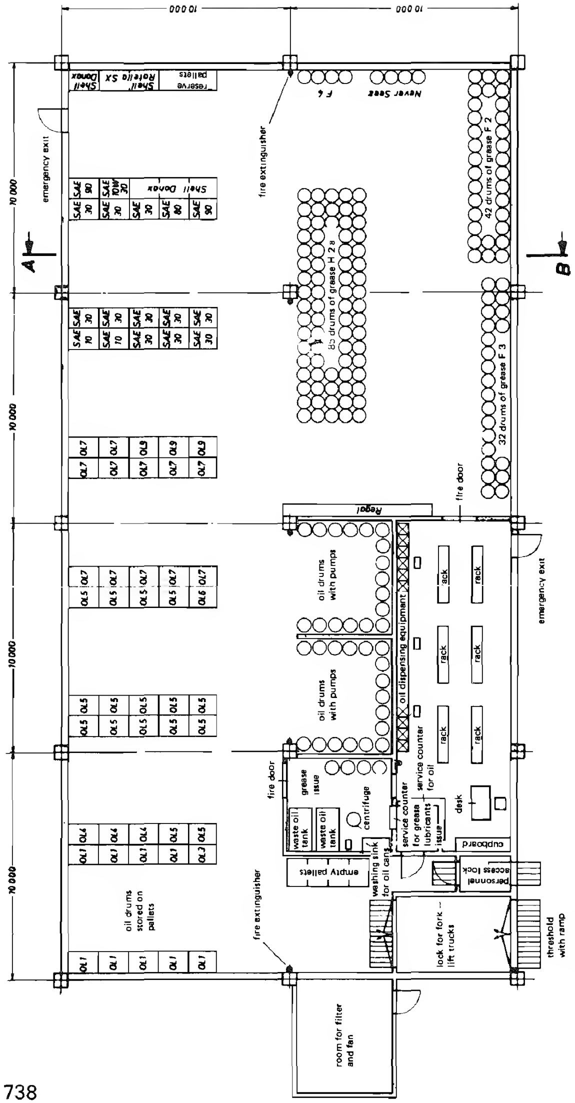


Fig. 4: Storage of lubricants

2.3 Method of storage

Storage facilities and arrangements will be geared to the questions: "How much? Where? How?"

2.3.1 Storage of drums

If a large number of drums has to be stored, they should be placed on timber supports (Fig. 5), on pallets (Fig. 6) or on racks (Fig. 7).

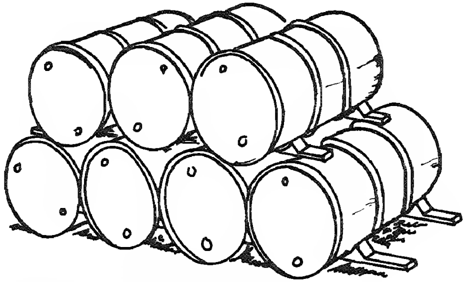


Fig. 5: Drums stored on timber supports

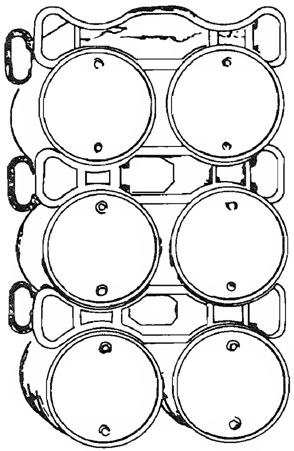


Fig. 6: Drums stored on pallets

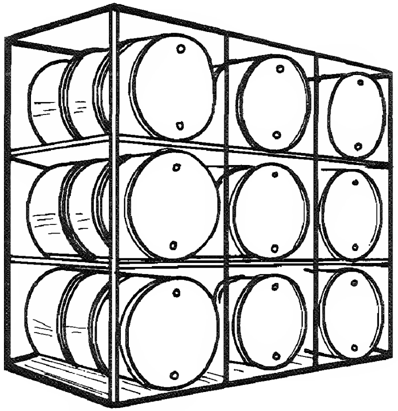


Fig. 7: Drums stored in racks

Storage in racks is the most suitable method from the point of view of using the lubricants in the order of their delivery to the store, whereas with stacked-up drums those deposited at the base of the stack will generally be used last.

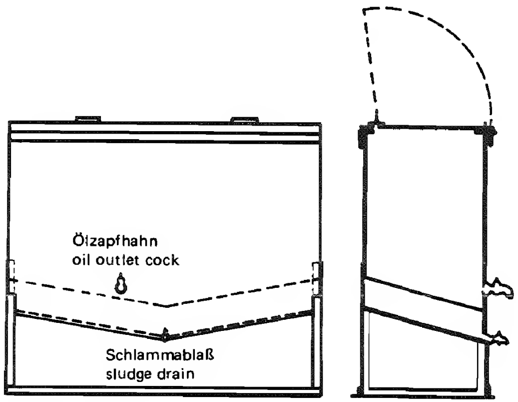


Fig. 8: Lubricant oil tank

2.3.2 Storage in tanks

Lubricating oil which is used in large quantities can most efficiently be stored in tanks. The size of these will depend on the rate of oil consumption in the cement works and on the quantities delivered to the works in each consignment (drums, tank vehicles, etc.). In any case the oil storage tanks should be of ample capacity so that, in the event of some days' delay in replenishment of supplies, no critical shortage will occur. Each tank should as a general principle always be used only for the same grade of oil.
Tanks installed inside buildings can be of simple welded construction (Fig. 8).
An important requirement is that the tank should have an inclined or a hopper-shaped bottom, so that moisture and impurities will settle at the lowest point and can be drained off there. For this reason, too, the tanks should be mounted sufficiently high above floor to allow a drum or other receptacle to be placed under the outlet.
All storage tanks should be provided with clearly legible markings or inscriptions. Dipsticks or other oil level checking devices are essential.
Outdoor tanks which are wholly or partly buried in the ground will have to conform to the statutory requirements for the prevention of ground-water pollution.

3 Issue of lubricants to consumers

Lubricants will be able to fulfil all the requirements only if they duly reach the lubricating points in an uncontaminated condition and not mixed with any quantity of some other grade or type of oil. To ensure this it is essential to have efficient arrangements for issuing the lubricants from the store. As a general principle, no lubricants should be issued except on production of a requisition note stating the quantity and brand of lubricant and the machine for which it is required. In the interests of economy, proper records of such data should be kept.

3.1 Issuing department

3.1 1 Location

In most cases it will be advantageous to issue the lubricants from the store itself. However, in fairly large cement works it may, under certain circumstances, be more appropriate to have several issuing points advantageously located in relation to the main consuming machinery. As a general rule, the lubricants must, on their way from the store to the machinery, be protected as much as possible from contamination with dust and dirt.

3.1.2 Size

The size of the lubricants issuing room will depend on the quantities and the number of grades or types of lubricant in regular use. Adequate space for installing and operating the dispensing equipment, the drilling oil mixers, the sinks and tables for washing and setting down oil cans, etc. should be provided. If necessary,

proper access for plant servicing trolleys and electrically powered trucks will have to be available.

3.1.3 Construction

What has been said on the subject of building construction in the section "Storage" is applicable also to the lubricants issuing and dispensing facilities. In cases where combustible liquids of category A, hazard class I – III, have to be handled, it is strongly advisable to accommodate these facilities in a separate room of appropriate design, so as not to be obliged to use flame-proof (explosion-proof) enclosure for all the pump motors, switchgear, etc.

3.2 Dispensing equipment

As a rule, the lubricants are dispensed by means of pumps which extract them from the original drums or through permanent pipelines from the storage tanks. One important object of these arrangements is to avoid as far as possible any transferring of lubricants from one receptable to another with the attendant risk of contamination. Depending on the requirements of the cement works and on the available facilities, one or more of the following possible methods of dispensing lubricants may be employed

(a) with manual equipment·
- oil from the original drums,
- oil from tanks;
- issue of grease:

(b) with electric equipment·
- oil from the original drums,
- oil from drums in dispensing cubicle;
- oil from dispensing cubicle, separate from pump with drum or tank;
- issue of grease.

4 Distribution of lubricants to the machines

Essential to economical lubrication management is that the distribution of lubricants to the various machines and installations must be linked to properly planned systematic maintenance. Indeed, the all-important rule of lubricants economy is that these operations must be correctly organized.
Maintenance of machinery comprises, in addition to cleaning, the servicing and lubrication of all the lubricating points, oil changing (including purging the system of used oil or filtering the oil for re-use), and remedial treatment of any faults that have begun to show up.
Planned maintenance moreover includes the scheduling of lubricating, oil changing and oil filtering intervals, with due regard to the plant operating conditions and requirements, as well as including preventive maintenance.
The success of these arrangements will depend to a great extent on the correct form of organization and the use of efficient maintenance equipment.

IV. Lubricants consumption

The consumption of lubricants will depend to a great extent on whether the cement works is relatively old, with aging machinery, or is a modern one with up-to-date equipment. Whereas open gear systems are still found in some older works, modern practice is to use enclosed gear units, while kiln and mill drives are of almost completely encased design.
The following approximate figures for guidance relate to a modern 3000 t/day cement works:

	consumption per t of cement	
	first year	subsequent years
oils of all grades	110 g/t	85 g/t
greases of all grades	35 g/t	30 g/t

A notable development that has been introduced into cement manufacturing machinery lubrication in recent years, more particularly for large drive units (kilns, mills, etc.), is the spray lubrication technique (Blanke, 1975; Wollhofen, 1975).

References

1. Blanke, H.-J.: Spruhhaftschmiertechnik in der Praxis. – In: ZKG 28/1975/392 – 397.
2. Wollhofen, G. P.· Sprühhaftschmierstoffe zur Antriebsschmierung in der Zementindustrie. – In: ZKG 28/1975/432 – 440.

O. Firefighting equipment

By B. Kohlhaas

Cement raw materials, clinker, cement and some auxiliary materials used in the manufacturing process are incombustible. All the same, outbreaks of fire in a cement works can by no means be ruled out. Office buildings, housing accommodation, laboratories and the lubricants store are subject to the normal fire hazards, while potentially disastrous major fires can occur in fuel stores and their ancillary installations. Besides, fires may start in electrical switchgear, cables, transformers, process control systems and measuring equipment.

Different methods and equipment are needed for dealing with different types of outbreak. For example, whereas some fires can be extinguished simply with water (fire class A), those occurring in fuel stores (coal, oil, gas) must be tackled only with special powder or carbon dioxide extinguishing agents (fire classes B and C). The same applies to outbreaks in electrical installations.

Another important distinction is whether the fire is in the open air (e.g., a coal stockpile) or inside a building.

A complete and effective firefighting system should therefore be obtained from, and in consultation with, an experienced specialist firm. It should also be checked whether, in the event of a serious fire, help from the public fire brigade can readily be obtained.

Since any major outbreak of fire is liable to cause loss of valuable materials and installations, it is essential to have adequate firefighting facilities in a cement works.

There is a wide range of firefighting equipment – from the simple hand extinguisher to sophisticated fire-engine type vehicles. Also, a distinction can be drawn between mobile and fixed apparatus.

For dealing with fires which can permissibly be extinguished with water it is generally sufficient to provide hydrants connected to a pipeline system fed by a pump delivering water under pressure. The hydrants, usually installed above floor level and equipped with hoses of sufficient length, are located at strategic points and are connected to the supply system, preferably comprising a ring main. Where cicumstances require this, the water for firefighting may be obtained from rivers or lakes in the vicinity or from ponds or basins constructed for the purpose.

All equipment should be regularly serviced and tested so as to be sure that it will function properly in an emergency. It is furthermore advisable to set up a works fire brigade, to train members of the personnel in firefighting and to organize fire drill at regular intervals.

Examples of various firefighting appliances are illustrated in Figs. 1 to 6.

Hand extinguishers (portable extinguishers)

Fig. 1a: Auto extinguisher with pressure gauge, filled with 2 kg of dry powder extinguishing agent, for fire classes A, B and C

Fig. 1b: Powder extinguisher, permanently charged with nitrogen for expelling the powder, 6 or 12 kg powder filling, for fire classes A, B and C

Fig. 1c: Powder extinguisher, 6 kg powder filling, internal gas cylinder for powder expulsion, for fire classes A, B and C

Mobile extinguishers

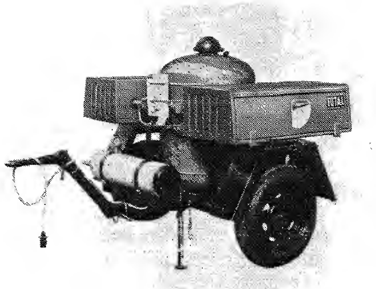


Fig. 2a: Powder extinguisher, 50 kg powder filling (foam compatible), for fire classes B and C, or 50 kg powder filling, for fire classes A, B and C

Fig. 2b: Powder extinguisher, mounted on trailer for towing by motor vehicle, 300 kg powder filling, for fire classes B and C, or 225 kg powder filling, for fire classes A, B and C

Fixed apparatus

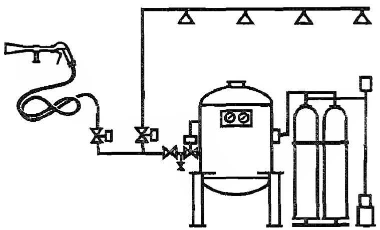


Fig. 3: Powder extinguishers for the protection of objects and premises, with fillings ranging from 50 to 15000 kg

Vehicle-mounted apparatus

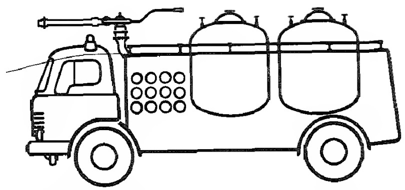


Fig. 4: Powder extinguisher vehicles of various types, with fillings ranging from 250 to 6000 kg

Fig. 5a: Water (gas pressure) extinguisher, with 10 litres water filling discharged by compressed nitrogen, for fire class A

Fig. 5b: Carbon dioxide extinguisher, filled with 2 kg of CO_2, with fog nozzle for fire class B or with gas discharge nozzle for classes B and C

Acknowledgements for illustrations
TOTAL-Foerstner & Co., Ladenburg/Cologne

Fig. 5c: Carbon dioxide extinguisher, filled with 6 kg of CO_2, with fog nozzle or with tube for CO_2 snow, for fire class B

Fig. 5d: Halon extinguisher, filled with 1.5 or 3.7 kg of inert vaporizing liquid, for fire classes B and C

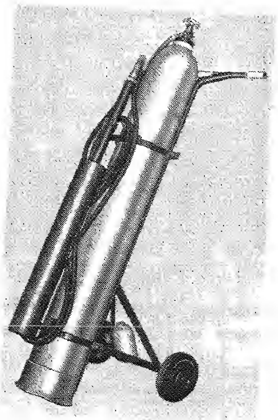


Fig. 6: Carbon dioxide extinguisher, mobile, filled with 30 kg of CO_2, for fire class B

P. Laboratory equipment

By B. Kohlhaas

I. Introduction

The laboratory in a cement works has a variety of duties to perform, including for example the examination of the raw material components, the decision as to which individual components are to be quarried and in what quantities, and the determination and monitoring of the raw meal composition. Moreover one of its most important tasks is the routine monitoring of the cement production process to ensure unvarying product quality, conformity to the relevant Standards, etc. Testing the accessory materials likewise comes within its sphere of activities. On the other hand, research is not in general conducted in the works laboratory. In order to meet the many and varied requirements, the laboratory must of course be appropriately designed and equipped. More particularly it should be so arranged that the various testing sequences can proceed in accordance with a logical pattern.

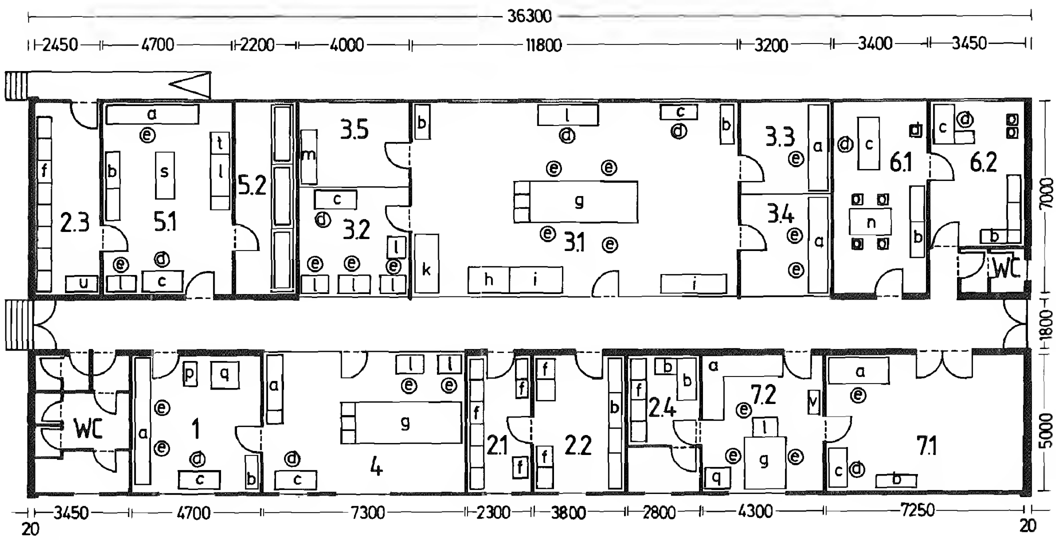


Fig. 1: Proposal for the layout and equipment of a complete cement works laboratory

Rooms

1 Preparation of samples
2.1 Store room for chemicals
2.2 Store room for laboratory equipment and accessories
2.3 Store room for technical laboratory
2.4 Store room for X-ray analysis equipment
3.1 Laboratory for chemical analysis
3.2 Weighing room (chemical balances)
3.3 Furnace and oven room
3.4 Room for physical measuring apparatus
3.5 Room for cleaning glassware, etc.
4 Shift laboratory
5.1 Technical laboratory
5.2 Room for curing of test specimens
6.1 Office for Head of laboratory
6.2 Writing room
7.1 Room for X-ray fluorescence analysis
7.2 Preparation room for X-ray analysis

Equipment and furnishings

a Wall work table
b Cupboard for apparatus or documents
c Writing desk
d Swivel chair
e Swivel stool
f Shelf
g Double work table
h Fume cupboard
i Work table
k Titration table
l Balance table
m Rinsing sink for cleaning glassware, etc.
n Table
o Chair
p Jaw crusher
q Disc mill
r Ion exchanger
s Compression testing machine
t Air-conditioned cabinet
u Large balance
v Tablet pressing machine (for X-ray analysis)

The information in this chapter is offered only as an example of the design and equipment of a modern laboratory, without laying claim to completeness in the treatment of this complex subject.
All the apparatus, chemicals, etc. should be available in the laboratory and be efficiently accommodated and stored. Suitable office furniture also forms part of the equipment to be provided. Fig. 1 is presented as a suggestion of what a modern laboratory building should comprise and how its internal layout may be. The individual work rooms should be of ample size and preferably all be on the same floor – the ground floor, if possible. If this is impracticable for architectural reasons, the rooms in question should at least be conveniently accessible from all parts of the cement works. Cellars or basements should not be used as laboratory space, but may be used for the storage of certain bulky materials or equipment (e. g., standard sand for mortar tests, empty receptacles and moulds, comparison specimens which have to be kept for some time, etc.) and may also accommodate the heating installation.

II. Proposed outline specification for equipment of individual rooms

General remarks

(a) All tiled work table tops are covered with red-brown matt-textured tiles, including bead edge tiles, jointed with acid-resistant mastic. The table tops have a reinforced concrete supporting slab and are precast in transportable units which are assembled in situ. Table top thickness about 60 mm.
(b) All plastic work table tops comprise a 38 mm thick supporting panel of flat-platen pressed high-density chipboard. The surfaces, including the edges, are faced with 1.3 mm melamine resin laminated plastic. The top surface with light grey pattern, the undersurface with reverse pattern. Table top thickness about 40 mm.
(c) The steel supporting frames consist of square-section (30 mm) tube, suitably stiffened and braced. Table legs are provided with level-adjusting screws.
(d) All base units and cupboards are made of flat-platen pressed chipboard, 19 mm thick, with plastic facing on both sides.
Base units are slid into the steel frames and should leave about 200 mm clearance above floor level.
Fume cupboards are supported on steel frames about 200 mm in height. The self-locking hinged doors have an opening angle of 100 degrees. Drawers are made of specially profiled and fully plastic-faced chipboard, with runner guideways and stops. Base units and fume hood superstructural parts are likewise of plastic-faced construction.
(e) Fittings and accessories conform to the relevant Standards and are coated with light grey plastic. They are provided with the necessary screw sockets.
(f) Sinks are made of acid-resistant brown-glazed or white-glazed stoneware and are provided with overflow, loose strainer, plug and trap.

1 Preparation of samples

1 wall work table (work bench)
size of table top: 4500 mm × 700 mm, height 900 mm above floor;
comprising more particularly:
1 tiled table top with cored tiles and apertures; the table top rests on:
1 tubular steel supporting frame, into which are slid:
2 base units, 600 mm wide, with 5 drawers one above the other;
3 base units, 900 mm wide, with 2 hinged doors on the front and internal shelf
1 base unit, 450 mm wide, with 5 drawers one above the other; furthermore:
2 end face panels; attachments in or on table top:
2 insertion funnels (145 mm × 145 mm);
2 supply columns for cold water, each with 1 outlet valve, 200 mm high;
2 supply columns for propane gas, with 2 valves;
2 supply columns for compressed air, with 2 valves;
at one end of table:
1 attached sink (600 mm × 400 mm × 300 mm) with bracket; behind this, on the tiled table top:
1 supply column for cold water, with 2 valves, 300 mm high.
1 cupboard for apparatus
size: 1171 mm × 494 mm × 1825 mm, with
2 leaf doors on the front, two-thirds glazed, with safety lock, with 4 adjustable internal shelves.
1 writing desk
size of desk top: 1500 mm × 750 mm, height 780 mm above floor, with:
1 table top of plastic;
base units from left to right:
1 base unit 450 mm wide with 4 drawers one above the other;
1 base unit 450 mm wide with 1 hinged door (right-hand mounted) on the front and internal shelf;
in addition:
2 end face panels.

2.1 Store room for chemicals

8 shelf units
basic unit 100 cm wide, 62 cm deep, 187.2 cm high, with 5 shelves.

2.2 Store room for laboratory equipment and accessories

4 cupboards for apparatus
size: 1171 mm × 494 mm × 1825 mm, with
2 glass sliding doors on the front, with safety lock, 3 adjustable internal shelves and 1 fixed shelf.
4 shelf units
basic unit 100 cm wide, 62 cm deep, 187.2 cm high, with 5 shelves.

2.3 Store room for technical laboratory

8 shelf units
basic unit 100 cm wide, 62 cm deep, 187.2 cm high, with 5 shelves

2.4 Store room for X-ray analysis equipment

2 cupboards for apparatus
size: 1171 mm × 494 mm × 1825 mm,
comprising inter alia:
2 glass sliding doors on the front, with safety lock, 3 adjustable internal shelves and 1 fixed shelf; the cupboards stand on steel supporting frames and are clear of the ground to provide foot space under them.
4 shelf units
basic unit 100 cm wide, 62 cm deep, 187.2 cm high, with 5 shelves.

3.1 Laboratory for chemical analysis

1 double work table
size of table top: 4050 mm × 1500 mm, height 900 mm above floor, with
1 tiled table top with cored tiles and apertures for sinks: base units, side A:
3 base units, 900 mm wide, with 2 hinged doors on the front and internal shelf;
2 base units, 600 mm wide, with 5 drawers one above the other; base units, side B:
3 base units, 900 mm wide, with 2 hinged doors on the front and internal shelf;
2 base units, 600 mm wide, with 5 drawers one above the other; furthermore.
2 end face panels;
attachments in or on table top:
1 rack for reagents, single-tier, length 3650 mm, width 300 mm, height about 350 mm, comprising:
4 supporting columns, each with 2 earthing-contact socket outlets for 220 V, also
2 mounting studs for stands or supports,
the working surfaces to be of wire glass,
in the tiled table top:
4 insertion funnels of acid-resistant brown glazed stoneware
(145 mm × 145 mm);
on the table top:
1 supply column for cold water, with 2 outlet valves, 200 mm high;
2 supply columns for cold water, each with 1 valves, 200 mm high;
2 supply columns for propane gas, each with 2 valves;
1 supply column for propane gas, with 4 valves;
2 supply columns for compressed air, each with 2 valves;
at one end of table:
1 sink made of acid-resistant brown glazed stoneware, 1450 mm × 390 mm × 275 mm, with:
1 basin (550 mm × 340 mm × 255 mm) and 2 suitably profiled lateral draining surfaces;
1 supply column for cold water, with 3 valves, 300 mm high.

1 fume cupboard
size of table top: 1500 mm × 750 mm, overall depth 850 mm, height 900 mm above floor, overall height of hood 2500 mm, with:
1 tiled table top with cored tiles and apertures;
the table top rests on:
1 base unit with double sliding door on the front and internal shelf;
furthermore:
2 end face panels;
in front, under the table top:
1 continuous boxing containing fittings and provided with appropriate apertures and holes for fittings, switches and plug sockets;
on the tiled table top:
1 hood superstructure with glazed side and rear walls;
on the front·
1 balanced sliding window, with counterweights, running on plastic rollers concealed in the columns;
all glazing to be of safety glass; the hood panels of wired glass; the hood rear wall with Eternit-Glasal (or similar) glazing material, with apertures for ventilation dampers; wooden components to comply with requirements stated in the "General remarks";
in rear wall:
2 ventilation dampers of grey PVC.
Fittings and accessories:
2 supply outlets for propane gas, each comprising:
1 supply column and 1 fume cupboard valve;
1 supply outlet for cold water, comprising:
1 supply column and 1 fume cupboard valve, also 1 insertion funnel of acid-resistant brown glazed stoneware (145 mm × 145 mm)
Attachments on fittings boxing:
2 earthing-contact socket outlets for 220 V;
1 switch for the fan,
1 switch for the lighting;
mounted over the hood (above the wire glass):
1 complete lighting assembly, including fluorescent tubes;
behind the rear wall of the hood:
1 suction duct of grey PVC (300 mm × 100 mm cross-section), beginning under the table top, with condensate trap;
duct over hood connected via transition piece to PVC duct leading to fan;
1 PVC pipe bend (90 degrees);
1 radial fan for laboratory use, complete with three-phase a. c. motor, impeller and casing of grey PVC; casing and drive assembly mounted on the same base, with shaft seal;
fan accessories;
2 collars;
2 fixing clips;
4 vibration dampers.

Note: If the fan cannot be installed in the room, it should be ascertained what length the air intake duct to the fan has or, alternatively, the precise location of the fan in relation to the exhaust discharge direction should be clearly established (a sketch will suffice). The height of the room should be considered in relation to these requirements.
The exhaust duct should be provided with a rain-excluding cowl made of PVC.

2 cupboards for apparatus
size: 1171 mm × 494 mm × 1825 mm, each cupboard having:
2 hinged doors on the front, 2/3 glazed, with safety lock,
4 adjustable internal shelves.
1 work table
size of table top: 2000 mm × 750 mm, height 800 mm above floor, with plastic top.
1 work table
size of table top: 1800 mm × 750 mm, height 800 mm above floor, with plastic top.
1 titrating table
size of table top: 2500 mm × 750 mm, height 900 mm above floor, total height 1900 mm, depth of top unit 200 mm with wooden top comprising Detopack covering, white colour, total thickness about 40 mm, edges protected with timber strips;
table top rests on:
1 steel supporting frame;
inserted into this are:
2 base units, 900 mm wide, with 2 hinged doors on the front and internal shelf;
1 base unit, 600 mm wide, with 1 hinged door, right-hand mounted, and internal shelf;
in addition:
2 end face panels;
on the table top:
1 light box, 2500 mm long, 1000 mm high, 200 mm deep, comprising 1 front frame, detachable, with translucent glass infilling, centrally divided;
in interior of light box:
1 complete fluorescent tube lighting system installed ready for connection.
1 writing desk
size of desk top: 1500 mm × 750 mm, height 780 mm above floor, with
1 plastic top;
base units from left to right:
1 base unit, 450 mm wide, with 4 drawers one above the other, with 1 tubular steel supporting frame, 1 sitting recess about 600 mm wide;
1 base unit, 450 mm wide, with 1 hinged door (right-hand mounted) on the front and internal shelf, with 1 tubular steel supporting frame;
in addition:
2 end face panels.
1 portable table
940 mm wide, 600 mm deep, 875 mm high, with
1 plastic table top;

1 tubular steel supporting frame with 2 pairs of square tubular legs fitted with rubber-surfaced castors (2 of which can be fixed); supporting frame has longitudinal bracings and 1 shelf.
2 swivel chairs
4 swivel stools

3.2 Weighing room (chemical balances)

3 weighing tables
size of table top. 900 mm × 620 mm, height 780 mm above floor;
comprising:
1 plastic top, provided with 1 aperture 460 mm × 460 mm for the weighing slab;
in the table top: 1 weighing slab 450 mm × 450 mm made of cast stone.
1 simple weighing table
size of table top: 1000 mm × 750 mm, height 800 mm above floor;
1 plastic top.
1 writing desk
size of top: 1500 mm × 750 mm, height 780 mm above floor, with
1 plastic top;
base units from left to right:
1 base unit, 450 mm wide, with 4 drawers one above the other, with 1 tubular steel supporting frame, 1 sitting recess about 600 mm wide;
1 base unit, 450 mm wide, with 1 hinged door (right-hand mounted) on the front and internal shelf, with 1 tubular steel supporting frame;
in addition:
2 end face panels.
1 swivel chair
4 swivel stools

3.3 Furnace and oven room

1 wall work table
size of table top: 3150 mm × 750 mm, height 900 mm above floor, with tiled top.
1 swivel stool

3.4 Room for physical measuring apparatus

1 wall work table
size of table top: 3150 mm × 750 mm, height 900 mm above floor, with
1 tiled top, comprising apertures and cored tiles;
1 tubular steel supporting frame;
2 base units, 900 mm wide, each with 2 hinged doors on the front and internal shelf;
1 base unit, 600 mm wide, with hinged doors on the front and with sink unit as integral feature;
1 base unit, 600 mm wide, with 5 drawers one above the other.
in addition:

2 end face panels;
in the tiled table top:
1 sink unit as integral feature, made of acid-resistant brown glazed stoneware (595 mm × 445 mm);
behind this, on the table top:
1 supply column for cold water, with 2 valves, 300 mm high;
1 supply column for propane gas, with 2 valves;
2 electricity supply columns, each with 1 earthing-contact socket outlet for 220 V.
2 swivel stools

3.5 Room for cleaning glassware, etc.

1 sink unit
size: 2000 mm long, 530 mm deep, height 900 mm above floor;
comprising:
1 double sink made of acid-resistant brown glazed stoneware (1060 mm × 530 mm × 240 mm)
for mounting on wall·
1 mixing valve unit for cold and hot water, with curved swivel outlet and screw connection for hose;
2 drainers.

4 Shift laboratory

1 double work table
size of table top· 4050 mm 1500 mm, height 900 mm above floor, with:
1 tiled top with apertures for sinks and cored tiles;
2 tubular steel supporting frames, with:
3 base units, side A, 900 mm wide, with 2 hinged doors on the front and internal shelf;
2 base units, 600 mm wide, with 5 drawers one above the other;
3 base units, side B, 900 mm wide, with 2 hinged doors on the front and internal shelf;
2 base units, 600 mm wide, with 5 shelves one above the other; in addition:
2 end face panels;
on the tiled table top:
1 rack for reagents, single-tier, length 3750 mm, width 300 mm, heigh about 350 mm, comprising:
4 supporting columns, each with 2 earthing-contact socket outlets for 220 V, also 2 mounting studs for stands or supports;
the supporting surfaces to be of wired glass mounted in a frame comprosed of rolled steel sections;
in the tiled table top:
4 insertion funnels of acid-resistant brown glazed stoneware (145 mm × 145 mm),

on the table top:
1 supply column for cold water, with 2 outlet valves, 200 mm high;
2 supply columns for cold water, each with 1 valve, 200 mm high;
2 supply columns for propane gas, each with 2 valves;
1 supply column for propane gas, with 4 valves;
2 supply columns for compressed air, each with 2 valves;
at one end of table:
1 sink made of acid-resistant brown glazed stoneware, 1450 mm × 390 mm × 275 mm, with:
1 basin (550 mm × 340 mm × 255 mm) and 2 suitably profiled lateral draining surfaces;
1 supply column for cold water, with 3 valves, 300 mm high.
1 wall work table
size of table top: 2400 mm × 750 mm, height 900 mm above floor, with:
1 tiled top with aperture and cored tiles;
table top rests on:
1 tubular steel supporting frame, inserted into which are.
1 base unit, 1200 mm wide, with 2 hinged doors on the front and internal shelf;
1 base unit, 450 mm wide, with 5 drawers one above the other;
1 base unit, 600 mm wide, with hinged doors on the front and recess for sink;
in addition:
2 end face panels;
in the tiled table top:
1 sink unit as integral feature, made of acid-resistant brown glazed stoneware (595 mm × 445 mm);
behind this, on the table top:
1 supply column for cold water, with 2 valves, 300 mm high;
1 supply column for propane gas, with 2 valves;
2 electricity supply columns, each with 1 earthing-contact socket outlet for 220 V.
1 writing desk
size of top: 1500 mm × 750 mm, height 780 mm above floor, with:
plastic top comprising:
1 460 mm × 460 mm aperture for weighing slab;
tubular steel supporting frame is fitted with 4 vibration dampers;
in the table top:
1 weighing slab 450 mm × 450 mm made of cast stone.
1 simple weighing table
size of table top: 1000 mm × 750 mm, height 800 above floor.
1 swivel chair
2 swivel stools

5.1 Technical laboratory

1 wall work table
size of table top. 3400 mm × 750 mm, height 780 mm above floor.

1 work table
size of table top: 2000 mm × 750 mm, height 900 mm above floor, with:
sheet-steel covering 3 mm thick on wooden backing panel, total thickness of table top 30 mm;
1 tubular steel supporting frame;
1 attached sink made of acid-resistant brown glazed stoneware (600 mm × 400 mm × 300 mm);
1 supply column for cold water, with 2 valves, 300 mm high.
2 cupboards for apparatus
size: 1171 mm × 494 mm × 1825 mm, each with:
2 hinged doors on the front, 2/3 glazed, with safety lock,
4 adjustable internal shelves.
1 simple weighing table
size of table top: 2500 mm × 750 mm, height 800 mm above floor, plastic top.
1 writing desk
size of top: 1500 mm × 750 mm, height 780 mm above floor, with:
1 plastic top and base units.
1 swivel chair
2 swivel stools

5.2 Room for curing test specimens

This room, with facilities for storing the specimens in water, should be air-conditioned and have no windows.

6.1 Office for Head of laboratory

2 book-cases, each approx. 120 cm wide
1 writing desk 180 cm × 90 cm
1 conference table 120 cm × 70 cm
1 swivel chair
5 chairs for conference table

6.2 Writing room

3 filing cabinets, approx. 120 cm wide
1 writing desk 156 cm × 78 cm
1 typing desk 120 cm × 50 cm
1 swivel chair
3 chairs

7.1 Room for X-ray fluorescence analysis

1 wall table with Resopal top
size of table top: 200 cm × 100 cm, without fittings or base units
1 cupboard for apparatus
size: 1171 mm × 494 mm × 1825 mm, with:
2 glass sliding doors on the front, with safety lock,

3 adjustable shelves and 1 fixed shelf in interior.
1 writing desk
size of top: 1500 mm × 750 mm, height 780 mm above floor, with:
plastic top;
base units from left to right:
1 base unit, 450 mm wide, with 4 drawers one above the other, also 1 tubular steel supporting frame, 1 seating recess about 600 mm wide;
1 base unit, 450 mm wide, with 1 hinged door (right-hand mounted) on the front, internal shelf, 1 tubular steel supporting frame;
in addition:
2 end face panels.
1 swivel chair
1 swivel stool

7.2 Preparation room for X-ray analysis

1 double work table with tiled top
size of table top: 200 cm × 150 cm, height 90 cm above floor;
3 electricity supply columns, each with two socket outlets and base units, but no other fittings.
1 sink unit
with mixing valve unit and joined to tiled-top work table without fittings, with base units, L-shaped comprising one leg of 2 m (sink) and one leg of 3 m (wall work table).
1 weighing table
size of table top: 900 mm × 620 mm, height 780 mm above floor, with.
1 plastic top provided with 1 aperture 460 mm × 460 mm for the weighing slab;
1 tubular steel supporting frame fitted with 4 vibration dampers;
in the table top:
1 weighing slab 450 mm × 450 mm made of cast stone.
3 swivel stools

III. Laboratory equipment with apparatus and measuring instruments

1 Preparation of samples

1 jaw crusher
for coarse reduction, feed opening 60 mm × 60 mm, discharge opening setting 1 to 20 mm, withdrawable crushing jaw, electric motor.
1 mill with grinding discs (toothed discs)
with 1.5 h. p. three-phase a. c. motor; cast steel, with toothed preliminary crushing screw, including 1 pair of special discs for fine grinding.
1 sample splitter with 50 mm passages.

1 sample splitter with 8 passages, each of 24 mm;
all parts of hot-dip galvanized steel sheet; receiving trays 8 litres capacity; dimensions of equipment 620 mm × 620 mm × 420 mm.
1 heavy mortar, high type, made of extra-hard special alloy, with pestle, non-machined, height 220 mm, top diameter 200 mm.
1 set of hand sieves
300 mm × 300 mm with wooden frame

1 sieve 31.5 mm square apertures
1 sieve 20.0 mm square apertures
1 sieve 16.0 mm square apertures
1 sieve 12.5 mm square apertures
1 sieve 10.0 mm square apertures
1 sieve 8.0 mm square apertures
1 sieve 6.3 mm square apertures
1 sieve 4.0 mm square apertures
2 sieves 2.0 mm wire cloth
2 sieves 1.0 mm wire cloth
2 sieves 0.5 mm wire cloth
2 sieves 0.25 mm wire cloth
2 sieves 0.20 mm wire cloth
2 sieves 0.125 mm wire cloth
2 sieves 0.09 mm wire cloth

1 air jet sieve, laboratory type, complete, with the following accessories:
test sieve drums, covering and frame made of V2A steel, 200 mm diameter, with attached sealing ring, DIN 4188:

5 of 0.032 mm aperture
2 of 0.040 mm aperture
5 of 0.063 mm aperture
10 of 0.090 mm aperture
5 of 0.125 mm aperture
5 of 0.200 mm aperture
1 laboratory vibrating disc mill
with sound insulation and time switch with 4 setting ranges (0–10 sec., 0–60 sec., 0–10 min., 0–60 min.),
together with:
1 hard alloy pot, 100 cm^3 capacity, with cover, lined with Widia (sintered carbide) alloy; grinding media entirely of Widia.

3.1 Laboratory for chemical analysis

1 ion exchanger, two-bed apparatus,
nominal capacity between two regenerations at 10° d foreign ion content 400 litres, effective rate 2.5 litres/minute;

accessories:
2 polyethylene containers, 5 litres capacity, for regenerating agents;
1 service water filter, size 12 K, for the separation of undissolved substances and suspended matter, with see-through plastic cover;
2 comparison manometers, 63 mm diameter, with fine division and throttle.
1 infrared heating bath (complete)
3 electric hot plates
approx. 300 mm × 450 mm; infinitely variable temperature control, up to about 350° C.
2 magnetic stirrers with heating
type RMH, with stand, electric cable, each with stirring rods 25 mm and 40 mm respectively.
1 rapid incinerator
for crucibles, up to 920° C, with time switch and heating element.
2 heating domes, 450 W each
support for round-bottomed flasks (1 litre flasks)
4 heating domes, 200 W each
support for round-bottomed flasks (250 ml flasks)
2 tripods and supporting rings
for 1 litre heating dome
4 tripods and supporting rings
four 250 ml heating dome
6 surface evaporators, 1000 W each
made of quartz, for the rapid evaporation and concentration of liquids, 200 mm diameter, with evaporating dish
6 stands for surface evaporators
4 platinum dishes (without covers)
each approx. 50 ml capacity, weight approx. 22 g
5 platinum dishes (with covers)
each approx. 30 ml capacity, weight approx. 29 g
1 water bath
755 mm × 260 mm, 220 V, with 4 openings
1 sand bath
450 mm × 300 mm, infinitely variable control, for 220 V, 50 Hz, 2 kW
1 vacuum pump
complete with drive

3.2 Weighing room (chemical balances)

1 automatic analytical balance
200 g capacity, 0.1 mg sensitivity, for 220 V, 50 Hz.
1 precision balance with digital scale
up to 3 kg, readings to the nearest 0.1 g.
1 mercury barometer
790 – 630 mm mercury column.

1 set of precision weights
from 1 mg to 100 g, brass, in case.
1 electronic desk top computer
with print-out of results.

3.3 Furnace and oven room

1 muffle furnace
with electronic temperature control adjustable up tp 1100° C, for short periods up to 1150° C; effective internal space. 170 mm wide, 90 mm high, 270 mm deep.
1 spare muffle for replacement
1 drying oven
rectangular type, electrically heated, adjustable from 40° to about 250° C; internal dimensions (approx.): 500 m wide, 478 mm deep, 500 mm high.
2 crucible furnaces, Simon Müller type, Model 3a with connection integral in furnace casing, with flex, for 850°C and 1000°C
1 chamber furnace
for temperatures up to 1500° C, heated by silicon carbide elements, effective internal space 120 mm × 100 mm × 320 mm, connected load 5.4 kW, for 380 V, 50 Hz, including:
4 complete sets of silicon carbide heating rods (12 rods per set).

3.4 Room for physical measuring apparatus

1 Eppendorf photometer, type 1101 M
(multiplier version), including Hg spectroscopic lamp, photomultiplier, 2 screening tubes, 3 diaphragms, 3 spare lamps, and connecting cable for 220 V a.c., including instructions for carrying out metal and water analysis.

accessories:
1 filter combination 623 nm
1 filter combination 578 nm
1 filter combination 546 nm
1 filter combination 492 nm
1 filter combination 405 nm
1 filter combination 436 nm
1 filter combination 365 nm

optical cells, type A.
2 cells 0.5 mm path length
2 cells 10 mm path length
2 cells 20 mm path length

optical cells, type B:
4 cells 10 mm path length
4 cells 20 mm path length
4 cells 40 mm path length

1 special cell holder for cells up to 40 mm path length
3 scale lighting lamps
3 micro cells (flow-through type) 10 mm
2 portable flue gas analysers
for the determination of CO_2, O_2 and CO with 4 absorption vessels
accessories:
12 gas collecting tubes with 2 cocks (1 ordinary and 1 capillary) capacity: 250 ml;
8 absorption vessels, Ströhlein-Krüppel type, with double washing action;
2 burettes, with Karlsruhe type stopcock, 0–21 per cent fine graduation, remainder with coarse graduation;
30 rubber bulbs, Scheibler type;
2 levelling bottles, 250 ml capacity.
1 flame photometer
for rapid emission photometric determination of alkalis and alkaline-earth metals; complete basic apparatus with accessories; furthermore:
2 acetylene cylinders, empty;
2 propane cylinders, empty;
1 set of calibration (standard) solutions,
1 photoelectric turbidimeter (nephelometer)
with pointer instrument and built-in voltage stabilizer, with alternative battery operation, complete with 2 optical cells (100 ml) and spare lamp 220 V.
1 water testing apparatus
for carrying out the following determinations:
total hardness, carbonate and non-carbonate values, all alkalinity values, phosphate content, pH, mineral acid, chloride content, inspissation, density of boiler feed water, content of dissolved oxygen, free and aggressive carbonic acid.
1 calorimeter
for determining the calorific value of oil, complete for 220 V, with steel cylinder (empty) for oxygen.
1 optical pyrometer (radiation pyrometer)
with 2 ranges of measurement (700° to 1500° C and 1200° to 2000° C), with Ni-Cd rechargeable battery, battery charger, and carrying case.
1 Prandtl-type pitot-static tube
3 mm opening, made of brass with brazed joints, for temperatures up to 400° C, length of barrel 500 mm.
1 inclined limb manometer

4 Shift laboratory

1 automatic analytical balance
200 g capacity, 0.1 mg sensitivity, for 220 V, 50 Hz.
1 precision balance
with digital scale, up to 3 kg, readings to the nearest 0.1 g.
1 electric hot plate
approx. 300 mm × 450 mm; infinitely variable temperature control, up to about 350° C.

1 magnetic stirrer with heating
comprising stand, electric cable, and 2 stirring rods (25 mm and 40 mm respectively).
1 Blaine air permeability testing apparatus (ASTM C 204-51) for determining the specific surface of powders, with permeability cell made of stainless steel, mounted on lacquered rough-service steel plate, with suction fan, filling funnel, 1 set of filter papers, 1 bottle of oil and 1 thermometer graduated in 0.1° C.
calibration sand (approx. 100 g), officially certified:
1 filling of sand I, specific surface approx. 2800 cm^2/g;
1 filling of sand II, specific surface approx. 4000 cm^2/g; 2000 Blaine filter papers 1.27 cm ‰.

5.1 Technical laboratory

1 precision balance, e. g., Sartorius model 2353
with digital scale up to 3 kg, readings to the nearest 0.1 g.
1 automatic balance
range 5 to 50 kg, scale divisions 100 g.
1 sliding-weight platform balance
made of steel, range 1.5 to 150 kg, scale divisions 100 g.
1 drying oven
rectangular type, electrically heated, settings from 40° to about 250° C.
1 Blaine air permeability testing apparatus (ASTM C 204-51)
for determining the specific surface of powders, complete.
calibration sand (approx. 100 g), officially certified:
1 filling of sand I, specific surface approx. 2800 cm^2/g;
1 filling of sand II, specific surface approx. 4000 cm^2/g;
2000 Blaine filter papers 1.27 cm ‰
1 electronic air permeability tester type 7207
for the rapid determination of characteristic values for process control purposes.
1 set of 2 calibration capillaries, with test certificate
1 air jet sieve, for laboratory use, complete.
1 air-conditioned cabinet
approx. capacity of test compartment: 500 dm^3;
temperature range +5° to +50° C, temperature accuracy ± 0.8° C; humidity range (measured on specimen) approx. 90 to almost 100 per cent.
1 Ultramat 070
standard apparatus;
temperature range 20° to 170° C, with built-in signalling clock and standard accessories.
2 thermohygrographs
with recording instrument (7-day record).

5.1.1 Technical laboratory (ASTM)

1 compression testing machine
type 2508/2560, class I, DIN 51220, max. test load 60 tonnes; adjustable test height by means of central spindle;

for the testing of cement prisms, lightweight concrete, refractory materials, lime, gypsum, natural stone (for certain tests special attachments will be required); powered by multi-plunger constant-delivery pump, with control unit, manometric force measuring system, type 0210;
1 load pacing control unit, type 02022;
1 loading rate sensor, type 02041;
1 extension column (100 mm).
12 two-piece cube moulds
made of brass, 50.8 mm edge dimension, conforming to ASTM C 109, type 2121.
12 clamping frames, type 2121.01.01
24 underplates, glass (120 mm × 120 mm)
1 compression testing unit
comprising 40.32 mm × 40.32 mm platens, for cement testing in accordance with ASTM C 349 with testing machine type 6313-04, furthermore:
1 pair of spare platens.
1 flexural testing machine, electrically powered for 630 kg maximum load, constant loading rate;

furthermore:
1 tensile testing unit, type 2701-32, confirming to ASTM C 190-63 for test specimens with 1 sq.in. cross-section.
12 moulds for making tensile test specimens (briquettes) 1 sq.in. ASTM C 180-58, type 2102.
12 clamping frames, type 2102.01.01
24 glass plates (120 mm × 120 mm)
1 flexural testing unit, type 2701-34, confirming to ASTM C 348-63 T
bearings ball-mounted, spacing 119 mm.
1 vibrating table
for the compaction of cement mortar and other specimens in triple mould according to the ISO-RILEM-CEM method;

Principle:
For compaction, two moulds are clamped to the stainless steel top of the table. The latter is then vibrated at a frequency of 3000 cycles/minute by a magnetic vibrator unit which is mounted in a sheet-steel casing, together with measuring and control equipment.
The table top, with dimensions of 560 mm × 355 mm, has a working height of about 900 mm and is mounted so as to vibrate freely in this casing. The total amplitude is adjusted by a control system to 0.6 ± 0.1 mm and is monitored by means of an indicating instrument. The run-up time to full amplitude is not more than 1 second. The required length of vibratory compaction time is pre-set by time switch. For use on 220 V, 50 Hz supply.
in addition:
quick-action clamping attachment, for fixing 1 or 2 triple moulds to the table.
1 vibrating table conforming to ASTM C 230
with mould, tamper and motor drive.
1 mortar mixer

with two mixing speeds (140 and 285 r.p.m.), conforming to RILEM-CEM, DIN 1164, ASTM, AFNOR and other Standards.
1 program device/basic unit
slide-in system, with logic elements, monitor lamps, connection to mortar mixer with multi-pin plugs;
at least the following slide-in program module is additionally required:

1 RILEM-CEM slide-in program module

60 sec., 140 r.p.m. (after 30 sec.: signal lamp "put in sand")
30 sec., 285 r.p.m.
15 sec., stopped (signal lamp "clean edge of mixing pan")
75 sec., stopped (period of rest)
60 sec., 285 r.p.m.

240 sec.

1 automatic sand feed device
The standard sand must be fed in at 30 seconds after the start of mixing and must then be added at a uniform rate of feed for 30 seconds. The sand is kept in readiness in a hopper on the device. At the required instant for starting the feed the sand outlet is automatically opened and remains open for about 30 seconds.
1 feed device
with mounting ring, for determining the vibrated bulk density of granular and powdered materials.
1 set of sieves conforming to ASTM
200 mm diameter, 50 mm height, with frame, bottom and cover of brass, and the following stainless steel woven-wire cloths: Nos. $3^1/_2$, 4, 8, 10, 16, 20, 30, 50, 100, 170, 200 and 325; furthermore: 1", $^3/_4$", $^1/_2$" and $^3/_8$".
test sieve drums
covering and frame made of V2A steel, 200 mm diameter, with attached sealing ring, ASTM:
2 of 0.038 mm aperture, ASTM 400
2 of 0.045 mm aperture, ASTM 325
2 of 0.063 mm aperture, ASTM 230

1 Vicat needle apparatus
complete with accessories, for determining the setting time in accordance with ASTM C 187; also 3 needles and 3 plungers;
in addition (as spares):
12 ebonite rings for setting determination in accordance with ASTM.
1 laboratory high-pressure autoclave, without agitator equipment for testing the soundness of cement in accordance with ASTM C 151 under saturated steam up to 25 atm. pressure; capacity about 8 litres; complete with fittings, thermal insulation, heating unit, cooling system, and the following additional equipment:
1 prism holder for 8 test prisms (1" × 1" × $11^1/_4$");
9 double moulds for making test prisms (1" × 1" × 10");

1 mixing blade, stainless;
1 shrinkage measuring device with measuring attachment for 1" × 1" × $11^1/_4$" test prisms (ASTM), including accessories, accuracy 0.01 mm, checking rod.
100 measuring studs (ASTM)
12 triple moulds
40 mm × 40 mm × 160 mm, confirming to RILEM-CEM, DIN 1164, ASTM C 348-63 T and MF 15-413;
4 three-piece top extension units
2 strike-off screeds (scraper bars for excess material removal)
2 demoulding attachments
4 cover frames
2 tampers for flexural test specimens, 150 mm × 20 mm, 0.7 kg;
2 tampers for shrinkage test specimens, 110 mm × 20 mm, 0.5 kg;
2 tampers confirming to ASTM, $^7/_8$" × $3^1/_4$";
500 kg of ASTM standard sand.

5.1.2 Technical laboratory (BS)

1 compression testing machine
type 1120/600, 600 kN (60 tonnes), class II, DIN 51220; for the testing of 100 mm cement prisms; for connection to: measuring and control console type 011019", with manual load pacing; load measurement and indication through two precision pressure gauges (class 0.6/250 mm diameter);
measuring ranges: 60 to 600 kN (6 to 60 t); 20 to 200 kN (2 to 20 t)
including adapters for 100 mm cubes and for 2.78" cubes;
in addition:
table for installing under the testing frame.
1 vibrating table
for the compaction of compression test cubes, 220 V, 50 Hz.
24 moulds for mortar cubes
2.78", complete with base plate.
500 kg of calibrating sand conforming to BS
20 Le Chatelier soundness testers
1 Le Chatelier bath
24 cube moulds conforming to BS 1881
100 mm edge dimension
5 spatulas BS 12
3 tampers BS 1881
4 spatulas BS 12 (weight max. 210 g)
1 slump test equipment set, conforming to BS 1881, Part 2 comprising:
slump cone, tamper, steel
strike-off screed, base plate and spatula.
1 set of test sieves conforming to BS 410
200 mm diameter, 50 mm height, with the following apertures: 0.045, 0.063, 0.09, 0.125, 0.25, 0.5, 1.0, 2.0 and 4.0 mm; furthermore, conforming to DIN 4188: 5, 10 and 20 mm; with cover and collecting pan.

rectangular screens
aperture sizes: 20 mm, 50 mm and 0.075 mm; 30 cm × 30 cm, with cover, collecting pan and suspension chains.
1 mortar mixer (Hobart mixer)
type: n 50, with Cr-Ni steel stirrer and 5 litres capacity.
1 Vicat needle apparatus conforming to BS
complete with accessories;
furthermore (as spares):
12 brass moulds

6.2 Writing room

1 electric typewriter

7.2 Preparation room for X-ray analysis

1 tablet press
5000 aluminium capsules, 38/39 mm diameter
1 sample splitter
splitter head with 8 passages of 25 mm; all parts made of hot-dip galvanized sheet steel; collecting trays 8 litres capacity; dimensions of the apparatus 620 mm × 260 mm × 420 mm.
1 laboratory vibrating disc mill, type TS 100
with sound insulation and time switch with 4 settings: 0–10 sec., 0–60 sec., 0–10 min., 0–60 min.;
for use on 220/380 V, 50 Hz;
with:
1 hard alloy pot
100 cm^3 capacity, with cover, lined with Widia (sintered carbide) alloy; grinding media entirely of Widia.
1 sieving machine
for 200 mm diameter test sieves, with time switch up to 60 min., for use on 220 V a.c.;
furthermore:
test sieve drums, DIN 4188, covering and frame made of V2A steel, 200 mm diameter, with attached sealing ring:

2 of 0.032 mm aperture
2 of 0.063 mm aperture
2 of 0.090 mm aperture
2 of 0.125 mm aperture
2 of 0.200 mm aperture
1 of 0.500 mm aperture
1 of 1.000 mm aperture

1 high-speed micro mill
MS 50, complete with accessories
1 tablet fusion apparatus

complete with accessories
1 automatic analytical balance
200 g capacity, 0.1 mg sensitivity, for 220 V, 50 Hz.
1 precision balance
with digital scale up to 3 kg, 0.1 g sensitivity
1 muffle furnace
with electronic temperature control system, adjustable up to 1100° C, for short periods up to 1150° C; effective internal space: 170 mm wide, 90 mm high, 270 mm deep.

IV. General laboratory apparatus

1 Bottles and boxes

1 permanently labelled bottle, white, 500 ml, with stopper, for each of the following:
hydrochloric acid 1:1 conc., sulphuric acid conc., ammonia conc., acetic acid conc., barium chloride 10 per cent, ammonium phosphate 10 per cent (6 bottles in all)
1 permanently labelled bottle, white, 250 ml, with stopper, for each of the following:
hydrochloric acid conc., sulphuric acid conc., ammonia conc., acetic acid 1:3 (4 bottles in all).
1 permanently labelled bottle, brown, 500 ml, with stopper, for each of the following:
nitric acid conc., hydrogen peroxide, potassium permanganate 3 per cent (3 bottles in all).
1 permanently labelled bottle, 250 ml,
for each of the following:
nitric acid conc., silver nitrate, methyl red (3 bottles in all).
12 narrow-necked bottles with glass stoppers, white, 500 ml
12 narrow-necked bottles with glass stoppers, brown, 500 ml
6 narrow-necked bottles with glass stoppers, white, 250 ml
6 narrow-necked bottles with glass stoppers, brown, 250 ml
6 dropping bottles, 100 ml
of brown glass, with flat caps
6 dropping bottles, 100 ml
of clear glass, with flat stoppers
10 wash bottles with supports, 1000 ml
Jena glass, with rubber stoppers
10 wash bottles with supports, 500 ml
Jena glass, with rubber stoppers
5 polyethylene wash bottles, 200 ml
10 polyethylene wash bottles, 500 ml
5 polyethylene dropping bottles, 50 ml
5 polyethylene dropping bottles, 100 ml

4 flat-bottomed bottles, 5 litres,
with bottom tube, rubber plug with bent glass tube (right-angled) and bulb tube
5 suction bottles (conical), with connection for rubber tube, 500 ml
5 suction bottles (conical), with connection for rubber tube, 1000 ml
3 suction bottles with tube, 0.5 litre
vacuum-tight, Jena glass, with 41 mm adapter, rubber sleeve 41 mm, rubber seal (Guko 42)
1 Woulfe bottle, 500 ml
2 stopcocks, attached laterally
10 plastic carboys, 25 litres, narrow-necked, with screw cap
10 plastic carboys, 10 litres, narrow-necked, with screw cap
20 polyethylene bottles, 500 ml, with narrow neck and screw cap
20 polyethylene bottles, 1000 ml, with narrow neck and screw cap
10 polyethylene bottles, 2000 ml, with narrow neck and screw cap
100 polyethylene powder bottles, 100 ml, wide mouth and screw cap
100 polyethylene powder bottles, 250 ml, wide mouth and screw cap
50 polyethylene powder bottles, 500 ml, wide mouth and screw cap
50 polyethylene powder bottles, 1000 ml, wide mouth and screw cap
3 round glass boxes, with loose cover,
150 mm diameter, 75 mm high, for filter paper

2 Beakers, flasks, cylinders and test tubes

36 beakers, Jena glass, 250 ml, low shape
24 beakers, Jena glass, 100 ml, low shape
24 beakers, Jena glass, 400 ml, low shape
24 beakers, Jena glass, 600 ml, low shape
24 beakers, Jena glass, 800 ml, low shape
24 beakers, Jena glass, 1000 ml, low shape
40 Erlenmeyer flasks, wide-necked, Jena glass, 300 ml
30 Erlenmeyer flasks, narrow-necked, Jena glass, 300 ml
30 Erlenmeyer flasks, narrow-necked, Jena glass, 500 ml
12 Erlenmeyer flasks, narrow-necked, Jena glass, 1000 ml
3 Erlenmeyer flasks, narrow-necked, Jena glass, 2000 ml
6 Erlenmeyer flasks, narrow-necked, with NS 29, 250 ml
30 flat-bottomed flasks, with flanged edge, medium length, 500 ml
30 flat-bottomed flasks, medium length, 250 ml
25 graduated flasks, Duran glass, with plastic stopper, 100 ml
25 graduated flasks, Duran glass, with plastic stopper, 50 ml
25 graduated flasks, Duran glass, with plastic stopper, 250 ml
25 graduated flasks, Duran glass, with plastic stopper, 500 ml
10 graduated flasks, Duran glass, with plastic stopper, 1000 ml
5 graduated flasks, officially calibrated, 50 ml
5 graduated flasks, officially calibrated, 100 ml
6 graduated flasks, Duran glass, with polyethylene stopper (calibratable), 100 ml
6 graduated flasks, Duran glass, with polyethylene stopper (calibratable), 200 ml

6 graduated flasks, Duran glass, with polyethylene stopper (calibratable), 250 ml
6 graduated flasks, Duran glass, with polyethylene stopper (calibratable), 500 ml
6 graduated flasks, Duran glass, with polyethylene stopper (calibratable), 1000 ml
6 graduated flasks, Duran glass, officially calibrated, 250 ml
6 graduated flasks, Duran glass, officially calibrated, 500 ml
6 measuring cylinders, high shape, 10 ml
6 measuring cylinders, high shape, 25 ml
6 measuring cylinders, high shape, 50 ml
6 measuring cylinders, high shape, 100 ml
3 measuring cylinders, high shape, 250 ml
3 measuring cylinders, high shape, 500 ml
3 measuring cylinders, high shape, 1000 ml
200 test tubes, 130 mm × 14 mm ext. dia.

3 Pipettes, burettes, titrating apparatus, pycnometers

10 measuring pipettes, clear glass, 0.1 ml
10 measuring pipettes, clear glass, 1.0 ml
10 measuring pipettes, clear glass, 2.0 ml
10 measuring pipettes, clear glass, 5.0 ml
10 measuring pipettes, clear glass, 10.0 ml
10 transfer pipettes, clear glass, 1 ml
10 transfer pipettes, clear glass, 2 ml
10 transfer pipettes, clear glass, 5 ml
10 transfer pipettes, clear glass, 10 ml
10 transfer pipettes, clear glass, 20 ml
10 transfer pipettes, clear glass, 50 ml
10 transfer pipettes, clear glass, 100 ml
5 Peleus bulbs
8 burettes
with Schellbach scale, lateral stopcock, 50 ml: 1/10, officially calibrated
3 burettes
calibratable, Schellbach, with straight stopcock and teflon plug, 10 ml: 1/50
3 burettes
calibratable, Schellbach, with straight stopcock and teflon plug, 50 ml: 1/10
3 burettes
calibratable, Schellbach, with straight stopcock and teflon plug, 25 ml: 1/10
2 burettes
calibratable, Schellbach, with straight stopcock and teflon plug, 10 ml: 1/20, brown glass
3 burettes
calibratable, Schellbach, with straight stopcock and teflon plug, 25 ml: 1/10, brown glass
4 automatic burettes
with lateral stopcock and Schellbach scale NS 29, 50 ml: 1/10, calibratable, with stock bottle (2 litres) and rubber bulb

4 automatic burettes
with lateral stopcock and Schellbach scale NS 29, 25 ml: 1/10, calibratable, with stock bottle (2 litres) and rubber bulb
4 automatic burettes
with lateral stopcock, without Schellbach scale, 5 ml: 1/50, brown, calibratable, with stock bottle (2 litres) and rubber bulb
4 spare burettes
with Schellbach reader NS 29, 50 ml: 1/10, calibratable
5 titrating apparatuses, Pellet type (calibratable)
bottle of wide shape (2 litres), with lateral outlet stopcock and automatic zero point adjustment, rubber bellows and stopcock between burette and bottle, burette with Schellbach scale, 50 ml: 1/10.
5 titrating apparatuses, Pellet type (calibratable)
bottle of wide shape (2 litres), with lateral outlet stopcock and automatic zero point adjustment, rubber bellows and stopcock between burette and bottle, burette with Schellbach scale, 10 ml: 1/20.
5 titrating apparatuses, Pellet type (calibratable)
bottle of wide shape (2 litres), with lateral outlet stopcock and automatic zero point adjustment, rubber bellows and stopcock between burette and bottle, burette with Schellbach scale, 10 ml. 1/50.
5 titrating apparatuses, Pellet type (calibratable)
bottle of wide shape (2 litres), with lateral outlet stopcock and automatic zero point adjustment, rubber bellows and stopcock between burette and bottle, burette with Schellbach scale, 25 ml: 1/10.
3 pycnometers
with capillary stopper, accurately adjusted, 50 ml
3 pycnometers
with capillary stopper, accurately adjusted, 25 ml
3 pycnometers
with capillary stopper and thermometer, adjusted, 50 ml
3 pycnometers
with capillary stopper and thermometer, adjusted, 25 ml

4 Watch and clock glasses, crucibles, dishes, funnels, filtering equipment

20 glasses, 30 mm diameter
20 glasses, 40 mm diameter
20 glasses, 50 mm diameter
20 glasses, 70 mm diameter
20 glasses, 60 mm diameter
20 glasses, 80 mm diameter
20 glasses, 100 mm diameter
20 glasses, 125 mm diameter
20 glasses, 150 mm diameter
25 porcelain crucibles C 102/1, 30 mm diameter

75 porcelain crucibles C 102/2, 34 mm diameter
25 porcelain crucibles C 102/3, 40 mm diameter
35 covers to C 102/1
10 covers to C 102/2
10 covers to C 102/3
10 sets of glass filtering crucibles, Jena glass,
of each of the following: 1D2, 1D3, ID4
10 iron crucibles, without cover, height 35 mm, diameter 45 mm
10 iron covers for these
10 pure nickel crucibles, height 35 mm, diameter 40 mm, wall 2 mm
10 covers for these
2 lead crucibles, height 40 mm, diameter 40 mm, wall 3 mm with cover
10 teflon crucibles, 25 ml
10 teflon beakers, with lip, 100 ml
6 porcelain dishes, shallow, 60 mm diameter
6 porcelain dishes, shallow, 100 mm diameter
6 porcelain dishes, semi-deep, 100 mm diameter, blue inside
6 porcelain dishes, semi-deep, 120 mm diameter, blue inside
10 sets of casseroles, diameters 100 mm, 125 mm and 150 mm with lip and handle
10 sets of porcelain evaporating dishes
semi-deep, with lip, diameters 80 mm, 90 mm, 100 mm, 185 mm, 230 mm
10 sets of porcelain evaporating dishes
shallow, diameters 80 mm, 140 mm, 200 mm
2 porcelain mortars, internally rough, with pestle, 160 mm diameter
2 porcelain mortars, internally rough, with pestle, 30 mm diameter
12 aluminium dishes, rectangular, 100 mm × 70 mm × 20 mm
24 ignition dishes, 60 mm × 95 mm × 15 mm
24 ignition dishes, 48 mm × 75 mm × 12 mm
40 analytical funnels, Jena glass, 110 mm diameter, rapid funnel
35 analytical funnels, Jena glass, 80 mm diameter, rapid funnel
5 Büchner funnels, glass, approx. 12 cm
3 glass funnels, 60° angle, 45 mm diameter
3 glass funnels, 60° angle, 80 mm diameter
3 glass funnels, 60° angle, 150 mm diameter
3 glass funnels, 60° angle, 250 mm diameter
10 filtering adapters
diameter 41 mm, with 10 rubber seals (GukO 42) for suction bottle and 10 rubber sleeves (41 mm) for filtering crucible

5 Glass tubes, glass rods, pinchcocks, tubes, tongs, spoons, spatulas, plugs, brushes, cloths

2 glass tubes
1 m long, right-angled bend at 900 mm, with outlet, 8 mm diameter, lower end with NS core 29, to fit Erlenmeyer flask 300 ml, tube extending down to about 15 mm from bottom of flask

10 kg of glass tubes, 4 to 10 mm diameter
20 tube connectors, of glass, for tubes of various diameters, 120 mm total length, 18 mm maximum diameter
10 glass tees, with connectors for rubber tubing, 8 mm diameter
20 U-tubes with ground-in stopcocks and side tubes 12.5 cm long
30 glass rods, 5 mm diameter, 130 mm length
30 glass rods, 5 mm diameter, 200 mm length
10 kg of glass rods, 3 mm to 6 mm diameter
6 pinchcocks, Mohr's type, 50 mm length
6 pinchcocks, Mohr's type, 60 mm length
6 pinchcocks, Mohr's type, 70 mm length
6 pinchcocks, Hoffmann's type, hinged, 17 mm wide
6 pinchcocks, Hoffmann's type, hinged, 20 mm wide
6 pinchcocks, Hoffmann's type, hinged, 30 mm wide
50 m of PVC tubing, 5 mm internal diameter
50 m of PVC tubing, 6 mm internal diameter
50 m of PVC tubing, 8 mm internal diameter
50 m of PVC tubing, 10 mm internal diameter
50 m of PVC tubing, 12 mm internal diameter
50 m of PVC tubing, 20 mm internal diameter
50 m of rubber tubing, red, 5 mm internal diameter, wall thickness 1.5 mm
50 m of rubber tubing, red, 8 mm internal diameter, wall thickness 2 mm
50 m of vacuum tubing, 5 mm internal diameter, wall thickness 5 mm
2 crucible tongs with platinum shoes, 1.5 to 2 g of platinum, length 50 cm
2 crucible tongs with platinum shoes, 1.5 to 2 g of platinum, length 25 cm
4 crucible tongs of 18/8 steel, length 20 cm
1 crucible tongs of 18/8 steel, length 50 cm
2 crucible tongs of pure nickel, length 20 cm
2 evaporating dish holders
6 weighing-in spoons with wooden or plastic handle
6 horn spoons, 140 mm length
5 spoons for chemicals, short-handled, 140 mm length
5 scoops of low-pressure polyethylene, 62.5 ml
5 scoops of low-pressure polyethylene, 150 ml
5 scoops of low-pressure polyethylene, 500 ml
5 scoops of aluminium
5 scoops of aluminium
5 ladles of special steel, 200 ml
10 large spatulas, 250 mm, with flexible stainless blade and wooden handle
15 weighing-in spatulas, with wooden handle, width 14 mm
5 spatulas of 18/8 steel, length 21 cm
2 forceps (blunt), length 120 mm
20 rubber wipers, spade-shaped
12 rubber wipers, rod-shaped
1 large assortment of cork bungs, bottom diameter from 7 to 45 mm
3 large assortments of rubber bungs, bottom diameter from 4 to 63 mm

5 sets of Suberit cork rings, for flasks, 8 mm, 11 mm, 14 mm and 17 mm diameter per set
5 spray brushes, 10 mm diameter
5 test tube brushes, wool-tipped, 15 mm diameter
5 test tube brushes, wool-tipped, 30 mm diameter
5 Erlenmeyer flasks and bottle brushes, 45 mm diameter
5 beaker brushes, with wooden handle, 60 mm diameter
5 rinsing brushes, 65 mm diameter
20 fine brushes, round tuft, approx. 200 mm length
20 fine brushes, flat tuft, 100 mm length
5 round-tufted brushes of bristle, approx. 30 mm diameter
10 soft cleaning cloths
20 cleaning rags
20 towels
50 glassware cloths
2 Kleenex towels
2 plastic holders to fit

6 Stands, supports and other auxiliary equipment

8 filter stands, quadruple, adjustable, made of PVC
2 test tube holders, for 12 tubes, made of PVC
4 pipette stands, for 24 pipettes, made of plastic
6 double burette holders, with plate support, 150 mm × 300 mm
6 tripods, 12 cm diameter
3 triangles of chrome-nickel wire, without tubes, leg length 60 mm
20 clay triangles 40 mm
20 clay triangles 70 mm
25 asbestos wire gauze pieces 16 cm × 16 cm
5 angled clips, without sleeve, 40 mm
3 support rings, without sleeve, 70 mm diameter
4 support rings, without sleeve, 100 mm diameter
10 sleeves for stands, 16 mm grip
20 double sleeves
5 stands with plates, supporting rod 750 mm, plate 125 mm × 200 m
5 supporting rods, 500 mm
5 supporting rods, 750 mm
5 supporting rods, 1000 mm
5 round clips, without sleeve, 25 mm
5 round clips, without sleeve, 40 mm
5 round clips, without sleeve, 60 mm
5 angled clips, without sleeve, 25 mm
2 lifting platforms, plate 200 mm × 200 mm

7 Desiccators, Kipp apparatus, water jet pumps, thermometers, hygrometers

2 desiccators
diameter 150 mm, with curved NS stopcock in cover, including porcelain plate
2 desiccators
diameter 200 mm, with curved NS stopcock in cover, including porcelain plate
3 desiccators
diameter 200 mm, with NS stopcock in side tube, including porcelain insert
3 desiccators
diameter 250 mm, with NS stopcock in side rube, including porcelain insert
1 Kipp gas generating apparatus
complete with NS stopcock 29/32 in bottom tube, capacity 1 litre
5 water jet pumps, with non-return valve
metal, $^1/_2$", stopcocks with smooth outlet tube
10 laboratory thermometers, 0 to 360° C
10 laboratory thermometers, 0 to 150° C
10 laboratory thermometers, up to 250° C, graduated in 1/1
5 general purpose thermometers, graduated up to 50° C
5 general purpose thermometers, graduated up to 100° C
5 general purpose thermometers, graduated up to 250° C
5 general purpose thermometers, graduated up to 420° C
1 window thermometer max./min., on glass plate, with magnet
1 hair hygrometer

8 Buckets, bowls, troughs, measuring vessels

5 plastic buckets, without lip and without cover, 10 litres
2 plastic buckets, with lip and without cover, 12 litres
5 round bowls, made of polypropylene, 200 mm diameter
5 round bowls, made of polypropylene, 335 mm diameter
5 round bowls, made of special steel, 335 mm diameter
5 rectangular bowls, ground and polished, approx. 250 mm × 180 mm
5 troughs, made of polypropylene, 320 mm × 250 mm × 135 mm
10 measuring vessels, with handle, made of polypropylene, 1 litre
Other equipment:
1 universal time switch
with programmable switching operation for 1-day period, on and off from 0.5 to 12 hours, 16 A, 220 V, 50 Hz
2 stop-watches
2 time interval meters, 60 minutes
2 time interval meters, 30 minutes
1 magnifying glass, 10 ×
1 iron mortar
standard mortar, tall shape, made of extra-hard special alloy, with pestle, 15 cm diameter, 18 cm height, unmachined

1 agate mortar
with pestle, external diameter 100 mm, standard quality
1 horseshoe magnet, 100 mm length
7 waste disposal buckets, with cover
2 air dryers
6 Bunsen burners for propane gas, with valve and pilot flame
2 spirit lamps, made of glass, 100 ml
2 blast lamps for propane gas
3 electric burners, 500 W
with Cr-Ni steel support for fairly large vessels and crucible holder of heat-resistant wire
3 gas lighters, each with 10 spare flints
5000 round filter papers, 12.5 cm, black ribbon
5000 round filter papers, 12.5 cm, white ribbon
5000 round filter papers, 12.5 cm, blue ribbon
5000 round filter papers, 12.5 cm red ribbon
500 folded filter papers, 32 cm diameter
25 weighing glasses, 30 mm × 50 mm, with ground-in class cover
3 weighing boats, made of aluminium, with counterweight, 8 cm length
20 refillable chinagraph pencils
in each of the following colours: red, green, black
30 boxes of spare leads, each containing 6 assorted leads
1 glass cutter of Widia steel with wooden handle and renewable cutting blade
1 cork borer set
1 cork borer sharpener
1 tool cabinet
complete, with usual tools for domestic use
3 tins of Vaseline
250 g of desiccator grease
1 large first aid kit for laboratories, fully equipped
5 pairs of rubber gloves
5 pairs of asbestos gloves
2 pneumatic eye bathing bottles
for immediate bathing of the eyes if splashed with caustic liquid
10 pairs of safety goggles
2 step-ladders

V. Chemicals

1 Inorganic Chemicals

50 × 2500 ml	hydrochloric acid, fuming, at least 37 per cent, A.R. (= analytical reagent)
4 × 2500 ml	sulphuric acid, 95 to 97 per cent, A.R.
10 × 2500 ml	nitric acid, at least 65 per cent, A.R. (1.40)
5 × 2500 ml	orthophosphoric acid, at least 85 per cent, A.R.

1 × 2500 ml	chromosulphuric acid for cleaning glassware
24 × 500 ml	hydrofluoric acid, at least 40 per cent, A.R.
1 × 2500 ml	perchloric acid, approx. 70 per cent, A.R. (approx. 1.67)
1 × 1000 ml	hydrobromic acid, at least 47 per cent, A.R. (approx. 1.50)
25 ampoules	Titrisol 0.1 N potassium permanganate
25 ampoules	Titrisol 0.1 N H_2SO_4
25 ampoules	Titrisol 0.1 N HCl
25 ampoules	Titrisol 0.1 N NaOH
10 ampoules	Titrisol 0.1 N oaxalic acid
25 ampoules	Titrisol 0.1 silver nitrate solution
50 ampoules	Titrisol 0.1 N Titriplex III solution
50 ampoules	Titrisol 0.1 ammonium rhodanide
2 × 1000 ml	Titriplex solution A 1 ml, for det. water hardness
2 × 1000 ml	Titriplex solution B 1 ml, for det. water hardness
1 × 250 ml	perhydrol (hydrogen peroxide), A.R.
20 × 5000 ml	ammonia, at least 25 per cent
1 × 1000 g	ammonium sulphate, A.R.
1 × 500 g	ammonium thiocyanate, A.R.
1 × 1000 g	ammonium peroxide disulphate, A.R.
1 × 500 g	ammonium nitrate, A.R.
1 × 500 g	ammonium iron(II) sulphate, A.R.
1 × 500 g	ammonium iron(III) sulphate, A.R.
4 × 250 g	ammonium heptamolybdate cryst.
1 × 500 g	ammonium carbaminate (carbonate), A.R.
4 × 500 g	ammonium acetate, A.R.
10 × 500 g	ammonium chloride, A.R.
10 × 250 g	ammonium oxalate, A.R.
5 × 1000 g	ammonium acid phosphate, A.R.
5 × 250 g	hydroxyl ammonium chloride, A.R.
4 × 500 g	barium chloride, A.R.
1 × 500 g	lead(II) acetate, neutral, A.R.
1 × 500 g	boric acid, cryst., A.R.
1 × 1000 g	cadmium acetate
3 × 250 g	calcium carbonate, precip. for analysis
4 × 500 g	calcium chloride dihydrate cryst., A.R.
2 × 1000 g	calcium chloride, A.R., medium fine, for drying
4 × 250 g	calcium chloride, A.R., for elementary analysis
2 × 1000 g	manganese(II) sulphate monohydrate, A.R.
1 × 250 g	potassium thiocyanate, A.R.
2 × 250 g	potassium iodide, A.R.
1 × 1000 g	potassium acetate, highest purity
1 × 500 g	potassium bromate, cryst., pure
1 × 250 g	potassium chromate, A.R.
1 × 500 g	potassium bichromate, A.R.
2 × 500 g	potassium carbonate, A.R.

1 × 500 g	potassium acid carbonate, fine-grained, A.R.
1 × 250 g	potassium peroxy disulphate, A.R.
4 × 250 g	potassium permanganate, A.R.
1 × 500 g	potassium sulphate, A.R.
4 × 500 g	potassium pyro-sulphate, A.R.
1 × 500 g	potassium chloride, A.R.
1 × 500 g	potassium nitrate, A.R.
1 × 1000 g	potassium nitrate, pure
1 × 500 g	potassium hexacyanoferrate(II), A.R.
10 × 1000 g	potassium hydroxide, in tablets, highest purity
1 × 500 g	potassium chlorate, A.R.
1 × 500 g	potassium cyanide, A.R.
10 × 1000 ml	caustic potash, approx. 50 per cent
2 × 250 g	iron(II) chloride, A. R.
2 × 250 g	iron(III) chloride, A.R.
2 × 1000 g	iron(II) sulphate
2 × 100 g	iron(III) oxide
1 × 250 g	mercury(I) chloride, A.R.
4 × 250 g	mercury(II) chloride, A.R.
6 × 250 g	tin(II) chloride
1 × 500 g	di-arsenic trioxide, sublimated, A.R.
2 × 25 g	Kupferron (N-nitroso-N-phenylhydroxylamine, ammonium salt)
4 × 25 g	silver nitrate, A.R.
2 × 100 g	zinc oxide, A.R.
10 × 250 g	copper(I) chloride, A.R.
1 × 250 g	copper(II) chloride, A.R.
1 × 250 g	magnesium chloride, A.R.
2 × 250 g	magnesium oxide, A.R.
1 × 500 g	magnesium sulphate, A.R.
2 × 1000 g	sodium acetate, highest purity
5 × 1000 g	sodium potassium carbonate, A.R.
1 × 1000 g	sodium acid carbonate, fine-grained, A.R.
2 × 1000 g	sodium carbonate, A.R.
1 × 500 g	sodium sulphite, anhydr., A.R.
1 × 1000 g	sodium sulphate decahydrate, cryst., A.R.
1 × 500 g	sodium peroxide, gran., A.R.
1 × 500 g	sodium thiosulphate, pentahydrate, A.R.
2 × 250 g	disodium acid phosphate, A.R.
1 × 500 g	tetrasodium diphosphate decahydrate, A.R.
1 × 100 ml	bromine, A.R.
1 × 100 g	iodine, sublimated, A.R.
2 × 1000 g	mercury, A.R., for analysis and for polarography
2 × 100 g	tin, grey, A.R.
2 × 1000 g	zinc, coarse powder, A.R. (for reducer fillings)
2 × 1000 g	marble, granulated, for carbon dioxide evolution

2 Organic Chemicals

5 × 2500 ml	acetic acid (glacial), at least 96 per cent, A.R. (approx. 1.06)
1 × 1000 ml	formic acid, 98 to 100 per cent, A.R.
1 × 500 g	oxalic acid, A.R.
3 × 1000 g	L(+) tartaric acid, A.R., cryst., highest purity
6 × 250 g	5-sulphosalicylic acid, A.R.
2 × 1000 ml	acetoacetic ester
2 × 1000 ml	2-propanol (isopropyl alcohol), A.R.
4 × 1000 ml	n-butanol (n-butyl alcohol), A.R.
12 × 2500 ml	ethanol (ethyl alcohol), approx. 95 per cent
3 × 1000 ml	diethyl ether, A.R.
2 × 1000 ml	acetone, A.R.
8 × 5000 ml	ethylene glycol, A.R.
4 × 2500 ml	m-xylene, highest purity
2 × 2500 ml	pyridine, highest purity
2 × 2500 ml	carbon tetrachloride, highest purity
2 × 2500 ml	chloroform, A.R.
1 × 500 ml	glycerin, double dist. (approx. 87 per cent), A.R.
2 × 500 g	hexamethylene tetramine, A.R.
1 × 1000 ml	formaldehyde solution, 35 per cent
1 × 250 g	starch, soluble, A.R.

3 Reagents

2 × 1000 g	Eschka's reagent, A.R.

4 Indicators

50 boxes	universal indicator paper pH 1–10
50 rolls	litmus paper, in plastic boxes, red
50 rolls	litmus paper, in plastic boxes, blue
1 × 25 g	phenolphthalein indicator
40 × 5 g	l-naphthol phthalein indicator
4 × 25 g	methyl orange indicator
1 × 500 g	indicator buffer tablets
40 × 25 g	calcon carboxylic acid indicator for metal titration
10 × 25 g	eriochrome black T, indicator for metal titration
1 × 25 g	bromo phenol blue indicator, A.R.
10 × 5 g	1,10-phenanthrolinium chloride, A.R., and redox indicator

5 Other Utilities

2 × 1000 g	sea sand, acid-cleaned and calcined, A.R.
2 × 1000 g	glass wool
4 × 1000 g	silica gel with moisture ind.
1 × 250 g	desiccator grease
2 × 250 g	soda asbestos, fine-grained, for elementary analysis
2 × 1000 g	asbestos for Gooch crucible
10 × 1000 ml	RCH CO absorbent
10 × 1000 ml	RCH O_2 absorbent

Subject Index

About this book

This new and completely rewritten edition of "Cement Engineers' Handbook" has been expanded to almost six times the size of the previous edition. It thus reflects the major developments and advanced sophistication that have characterized cement manufacturing methods and technology in the past decade or so.

The subject matter of the book as a whole substantially covers the range of problems which concern the technologist engaged in present-day cement manufacturing practice. All essential and important information has been brought together in this book, which thus goes a very long way towards answering all the questions with which its users are likely to be faced in the course of their duties.

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